NOTES

ON

LEAD AND COPPER SMELTING

AND

COPPER CONVERTING.

BY

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2 cop. rec'd

NEW YORK AND LONDON:
THE SCIENTIFIC PUBLISHING CO.,
1897.

PREFACE.

This book is precisely what is indicated by its title—a series of notes on the practical work in lead and copper smelting, including the converting of copper matte. It is by no means a treatise or an attempt at a treatise. No effort has been made to trace in order all the steps in beneficiating ores by smelting from the crude material to the marketable product. This has already been done ably by other writers. It has seemed, however, that the experience gained in the everyday operation of three large works, extending over a period of ten years, might be useful to others who are engaged in similar work. Progress in any art is helped by an interchange of ideas. Hence these notes are offered.

Acknowledgement is due Mr. Wm. Braden for the reproduction of drawings of the settlers used at the Arkansas Valley Smelting Works, Leadville, Colo., from his paper in the *Transactions of the American Institute of Mining Engineers*, Vol. XXVI; and to L. S. Austin for the illustrations of the matte-pots and slag-trucks employed at the Omaha & Grant Smelting Works, Denver, Colo., which are taken from his paper on the separation and disposal of slag in *The Engineering and Mining Journal* of November 23, 1895; also to Julius A. Dyblie and John Bendixen for valuable services in preparation of plans and drawings.

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CHAPTER I.

Copper Matte Smelting.

The matting of ores without the presence of copper to serve as a carrier to collect the silver has been attempted on many occasions, but has only resulted in failure, the losses being too large under ordinary conditions to admit of treatment in this way. A small percentage of copper can be used with success, but if the resulting matte runs below 5 per cent. copper it is very doubtful if the process can be made successful. In cases where a large tonnage of ore is to be treated with a small amount of copper, it would be advisable to crush and roast a part of the matte produced and smelt it along with the charge to supply the copper needed. But in cases where the ores are sulphides, without the presence of either copper or lead to act as a carrier, better results can be obtained by leaching in localities where the conditions will not admit of marketing the ore to custom smelters. The reasons are that the tonnage of matte produced is not easily transported, and when sold has to pay treatment charges as well as allow for losses in subsequent treatment. If it is possible by the concentration of the matte to save enough copper to overcome these objections, then it becomes a question of costs to determine which is the most economical, and each particular case is an independent proposition and should be treated accordingly.

Emergencies arise, and it may be necessary to convert a lead-smelting plant into a mixed one of lead and copper, or *vice versa.*

In 1890 at the works of the Arkansas Valley Smelting Co., in Leadville, copper ores of sufficient quantity to justify separate treatment were received, and accordingly two of the

old lead furnaces were changed to copper furnaces by the very simple process of filling up the lead crucible and placing an overflow pot under the slag-tap, which pot was later replaced by a forehearth of the type shown in Figs. 1 to 4.

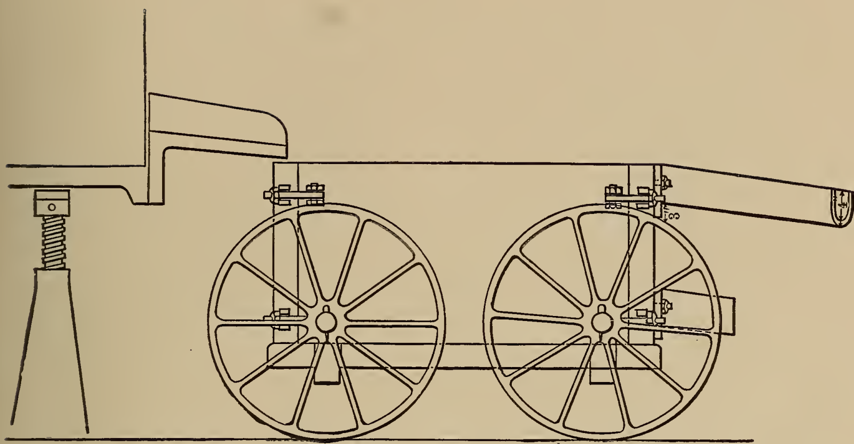
The running of these furnaces was at the time a novel feature in Leadville practice, as prior to that time all the blast-furnace work had been lead smelting and no strictly copper smelting had been done. The copper ores were mainly from the Maid of Erin, with a considerable quantity of iron sulphides from the Mahala and Wolftone mines.

The greater portion of the ores that could be used on the charge were sulphides of such a character as to require crushing and roasting, and, as a consequence, the charge was about 70 per cent. calcines, including roasted matte from the lead furnaces known as furnace or lead matte, which, however, contained about 7 per cent. copper, the result of concentration of small amounts of copper from the ores fed to the lead furnaces. This material was consequently fine enough to make an excessive quantity of flue dust and have a serious effect on the running of the furnace.

It might appear to the inexperienced that the size of material to be smelted would have no great effect on the cost of treatment, losses and tonnage of a furnace, but if any such persons should have to contend with the complications arising from an exceedingly fine charge, they would soon come to the belief that there is a very close relation between the size of the charge material within certain limits and the tonnage that can be put through.

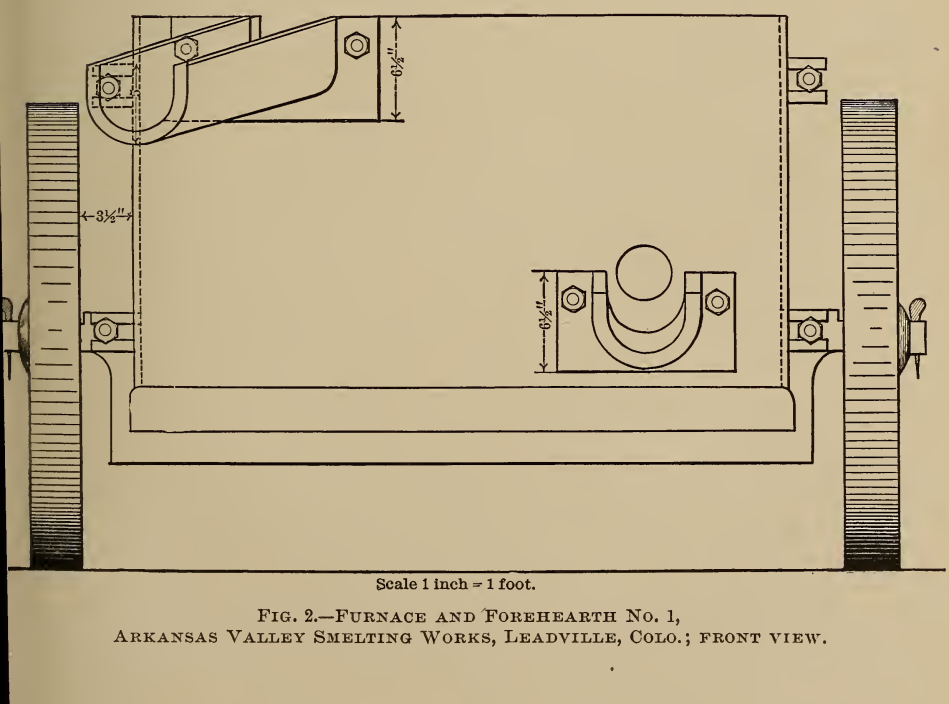
There was a twofold object in running the copper furnace; first, to treat separately the lead ores and such ores as contained no lead and some copper, and, second, to desilverize an accumulation of foul slag of long standing which previous administrations had left on hand. This slag was in part the bottoms of pots or that portion of the slag next to the layer of matte, and of old make, and the other part was the shells resulting from the Devereaux pots in use on the lead furnace, of which six were in blast most of the time.

It is due to state by way of explanation that lead was rather scarce in all the Colorado smelters from 1889 to 1891, and, as a consequence, the lead furnaces were running on a



Scale ½ inch = 1 foot.

FIG. 1.—FURNACE AND FOREHEARTH NO. 1,
ARKANSAS VALLEY SMELTING WORKS, LEADVILLE, COLO.



Scale 1 inch = 1 foot.

FIG. 2.—FURNACE AND FOREHEARTH NO. 1,
ARKANSAS VALLEY SMELTING WORKS, LEADVILLE, COLO.; FRONT VIEW.

short allowance of lead, 8 per cent. to 10 per cent. on the charge, with a long allowance of zinc and magnesia, and the conditions were such that the slag assays were seldom below 1.5 ounce Ag and more frequently 2 and 3 ounces Ag per ton. As a consequence of the high character of bullion—300 to 400 ounces—and low percentage of lead this was to be expected, and to remedy it as far as possible and to prevent an excessive silver loss the greater portion of the shells was fed into the copper furnaces. If it had not been for this addition of coarse, fusible material to the charge, the furnaces would have been even more difficult to manage than they were. Being only 36 by 84 inches, the furnace was not large enough to put through a tonnage that would keep a constant flow open, and, as a consequence, intermittent tapping, the same as on lead furnaces, was the practice. The slag composition attempted was 34 SiO_2, 33 FeO, 20 CaO, but owing to the feeding of so much slag of unknown and varying composition the analysis of the resulting slag showed up somewhat incorrectly owing to the presence of some lead, and did not permit of the formation of as clean slags as can be made when the charge contains no lead. It is a well-known fact that in matte smelting lead is scorified, and, owing to its strong affinity for silver, will carry it into the slag, whereas if only copper and silver are present the slags will be much cleaner.

The furnaces were arranged with a double tap in front, and every time the overflow pot was changed the lower tap was opened and the accumulation of matte in the furnace drawn off. The amount thus obtained would depend very much on the condition of the furnace, and ran from one to six or eight pots of clean matte before the slag would make its appearance. The condition of the furnace was mainly dependent on the amount of matte produced by the particular charge the furnace was running on.

In cases of reconcentration before shipment, when the matte production was very heavy, perhaps 40 per cent. of the charge, the furnace would cut out all accumulations of crust in the shaft and below the slag-tap and get into excellent condition. On the same composition of slag and percentage of fuel, with the regular charge on, where the matte produc-

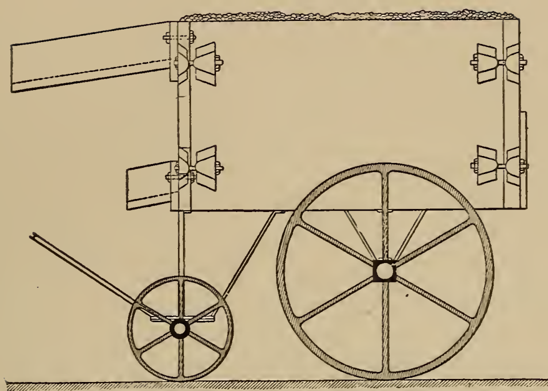


FIG. 3.—FOREHEARTH, FURNACES NOS. 2 AND 3,
ARKANSAS VALLEY SMELTING WORKS, LEADVILLE, COLO.

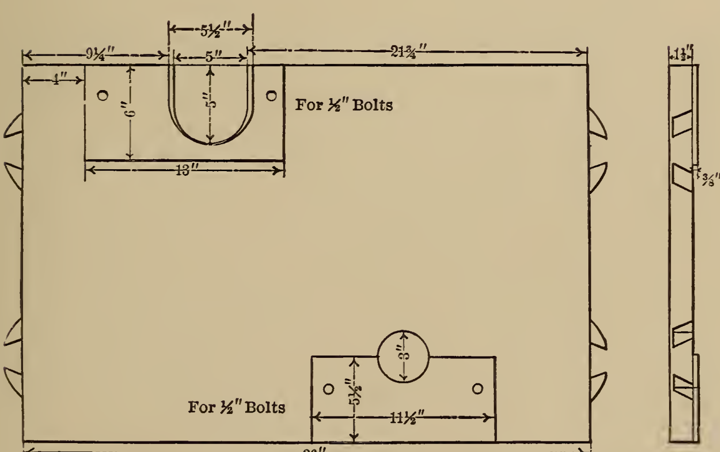


FIG. 4.—FOREHEARTH, FURNACES NOS. 2 AND 3,
ARKANSAS VALLEY SMELTING WORKS, LEADVILLE, COLO.

tion was necessarily small to effect a concentration of ten into one or more, the zone of fusion would travel up and the bottom of the furnace become crusted to such an extent that the crucible would hold not more than one pot of matte between the slag and matte-taps.

As a consequence of small matte production and high concentration, it frequently happened that the matte-tap could not be opened until a charge was put on the furnace that would produce more matte, either by making lower grade or by feeding back some of the matte already produced.

As a general thing less than 15 per cent. matte production in a strictly matting furnace will result in the tuyeres becoming hard and black and the rising of the zone of fusion to the top of the charge, which, of course, results in the fuel being consumed before it should, and the consequent loss of the heat escaping up the stack, as well as the inevitable loss of silver that will follow smelting in a furnace hot on top. Many furnaces have been, and doubtless are now, run on less matte production, but to run steadily at its maximum capacity is what is expected of a well-behaved furnace, and in order to do this it is essential to have the matte production in excess of rather than below 15 per cent. Likewise the production of matte in a lead furnace has a very marked influence on the minimum amount of lead that can be used on the charge. For example, it might be quite possible to run successfully on as low as 7 or 8 per cent. of lead, reinforced by a heavy production of matte, and altogether unprofitable to attempt it without the matte. Unless the corrosive effect of something beside slag is acting in the furnace the cold blast will chill the slag, resulting in crusts and the elevating of the zone of fusion. With the zone of fusion high above the tuyeres in a lead furnace the losses by volatilization become higher until the maximum is reached, when the entire amount is lost.

To return to the subject in hand, the handling of small furnaces with intermittent slag-tap and with matte production as low as 10 per cent. or lower, it must be said that it requires a very careful handling on the part of the furnace-men, even allowing that the slags are of the best possible composition. To keep the zone of fusion down at the tuyeres

is the essential condition of success, and to attain this is not entirely in the hands of the man who figures the charges. The charge as it comes to the furnace may be either too fine or too coarse to obtain the best results, and while in the first case nothing but the coarsest of coke should be used, in the latter the use of a sledge hammer on the large pieces will materially improve matters.

The proper placement of the charge in the furnace is a matter of the greatest importance, and it is hard to decide where the more experience is required, on the feed floor or in handling the furnace below.

In a case where the furnace temporarily becomes cold, as is likely to happen through the variation in matte production, it may become advisable to increase the fuel or put back a charge of straight matte in order to repair the threatened bad effect, and if the feeder does not know his business, or if the furnaceman does not properly attend to his duties, the result may be slag in the tuyeres, a disagreeable experience with sledge hammers for a time, and what is known as muscular metallurgy.

When a furnace has from any cause, be it fine ore, a bad slag, small matte production, or any other fault, become dark at the tuyeres and hot on top, to know the quickest way in which to remedy the difficulty is what is required of the man in charge. If it is a copper furnace the easiest way is by putting on matte charges until the tuyeres get bright and the crusts or accretions have disappeared sufficiently so as to warrant the matte charges being taken off or reduced in number by putting on one matte charge to one, two, or three, or more, of ore charges. It frequently happens at shift change, especially early in the morning when the night men are tired and are doing their work in a somewhat lazy and careless fashion, that the furnace is allowed to run down and get very hot on top. At such time the operation of the furnace assumes the character of pyritic smelting, the oxidation of sulphur being much greater than when the furnace is fed at its proper level, with the result that the matte production in proportion to the charge falls off rapidly and consequently increases in grade of copper contents. The excessive loss of

heat up the stack results in the furnace running cold, and when suddenly filled up again the consequence is bad running and probable freeze-up.

It is not the purpose of the writer to refer only to ideal conditions where everybody connected with the furnace knows his business and attends to it properly, but to take from the experience of the past that which may be valuable to others in the future. Too much blast will result in driving the fire to the top of the charge, and, providing the furnace will stand it, a reduction of the blast will assist the matte charges in bringing the zone of fusion down. On the other hand, tonnage must be maintained, and in order to make a furnace run on a good charge above 100 tons per day it is necessary to drive the wind into it at a lively rate, and a No. 7 Roots blower will have to make 140 revolutions per minute to furnish blast enough. This amount of blast would be all right for a copper furnace 42 by 120 inches at the tuyeres, but would be too much for a smaller furnace on the same charge, and entirely too much for a lead furnace of the same size.

It is natural to discuss at this point what is the approximate amount of wind to blow in in order to obtain the best results in copper and lead smelting. Taking the listed displacement of a Roots No. 7 blower at 65 cubic feet per revolution, and allowing 140 revolutions per minute for a copper furnace 42 by 120 feet at the tuyere, we get 9,100 cubic feet per minute for a furnace with cross section of 35 square feet, or 26 cubic feet of blast to each square foot of furnace section. Under ordinary conditions 70 revolutions per minute would be about the proper amount for a lead furnace of the same size to run on and would give 13 cubic feet per square foot of furnace section.

These figures are of necessity only for normal conditions of charge and tonnage, and if conditions arise for greater or less tonnage or to reduce the loss by volatilization, the amount of blast supplied has to be regulated to suit those conditions.

Also, the blowers may through wear become leaky to such an extent that they will not deliver the calculated amount of air to the revolution, and, as a consequence, would have to be run faster. I have never personally tested my blowers to find out how much they would deliver per revolution and at

varying pressure, except at Anaconda, where a No. 4 Roots was run with a closed outlet at 20 revolutions per minute and gave a pressure of four pounds to the square inch in the blast pipe, which would mean that if run against a pressure of four pounds it would deliver no air at all, or, in other words, the leakage at four pounds was 100 per cent.

With Baker blowers the leakage is much higher, and I do not believe that they could be made to generate a pressure of more than two pounds before the leakage would be 100 per cent. This would indicate what experience is slowly teaching all blast furnacemen—that the efficiency of Baker blowers, or blowers of that type, is much less at any pressure than blowers of the Roots type. Fifteen years ago, when Leadville was a new camp and when the writer received his first baptism of fire in the smelting business, there was not a Roots blower in any of the smelters that were in operation, and this tendency to oppose change has been maintained to a great degree even in the construction of new plants. It is an easily demonstrated fact that if the efficiency is lower the power required to deliver the blast to the furnaces must necessarily be larger than would be the case if blowers of the Roots type were used. As there are now two competing firms building this class of positive blowers, the writer can not be accused of unjust prejudice.

The number of moving parts and the closer contact required for a more positive delivery of the air, are the reasons for the greater efficiency and consequent lower consumption of power. The difference in power consumption will amount to fully 50 per cent. in favor of the Roots type, and regardless of first cost would soon pay for entirely new blower plants for many works now in operation. The listed speed for a No. 7½ Baker is only half that of a No. 7 Roots of the same displacement per revolution, so that on a basis of cost alone one Roots type is worth two Bakers, and on a basis of operation the ratio is two to one in favor of the Roots type, so that finally it would appear that the relative values of the two types is about four to one in favor of the Roots.

The type of furnace has also much to do with the amount of blast required for successful running. If, for instance, it is a constant discharge, the flow of slag must be rapid enough

to keep the slag-spout open, and this can not be done on much less than 60 tons of furnace charge per day. If the furnace is small and the character of charge such as not to admit of rapid smelting, it may be better to run an intermittent tap with a moderate blast than to try forcing the tonnage with high blast and run the risk of driving the fire up and running the furnace cold at the tuyeres.

These were the conditions at Leadville. The ore was fine, the matte production small, and the tonnage that could be handled not enough to keep a constant discharge open. A forehearth of the Herreshoff type with water-jacketed sides was constructed, but when the furnace was started it was found that the cooling effect of the water was too great and the entire contents of the hearth soon became solid. The hearth was changed and the water-jackets allowed to remain empty with better results, but the neck connecting the hearth with the furnace would become closed and slag would be forced out of the tuyeres after about twelve hours' run, and it was necessary to change the hearth at least that often.

The hearth was then constructed of cast plates with a pipe cast around the hole which connected with the furnace, and a similar plate with water circulating around the hole in the furnace front. This construction worked satisfactorily as long as the matte production was small, but if as much as 20 per cent. matte was produced it would cut out and discharge the contents of the furnace on the floor. This type of furnace was not adapted to steady running under varying conditions, since the changing of the hearth necessitated a shutdown which was unavoidable and frequently of an hour's duration.

After running some time under the conditions stated, with a matte production of about 10 per cent. and varying in grade from 15 to 40 per cent. copper, and in silver from 150 to 600 ounces, the experiment was tried of mixing in a small percentage of speiss with the ore before roasting. At first the quantity used was one-fifth of the calciner charges, and as no bad effect was observed on the calciners or blast furnace the proportion was increased until one-half of the calciner charges was arsenical speiss that had been produced several years before and was regarded as a waste product.

This speiss contained an average of 18 ounces Ag, one-tenth

ounce Au, 2 to 5 per cent. Pb, 15 to 20 As, about the same amount of sulphur, and the remainder iron. It had been made when the ores of the camp were chiefly of carbonate or oxidized character, and was a great annoyance at the time of its production as well as entailing a heavy loss of silver and gold in smelting. The accumulation of ten years' operations was stacked up in one corner of the slag-dump in the hope that by some undiscovered process it might some day be treated and the silver and gold recovered. There were about 3000 tons of it at this time on the dumps of the Arkansas Valley works, representing a large sum of money in gold and silver as well as the value of the iron excess as a flux for silica. At first this experiment did not promise to turn out well, as only the finest portions of the old pile could be used, and it was thought that when they were consumed the golden eggs would be gone. It will be understood that the crushing of speiss in lumps, varying in weight from 20 to 80 pounds and of a hemispherical shape, is attended with great risk of breakage to the machinery, and especially when there is some metallic lead to be found associated with it. The speiss was therefore thoroughly culled before sending it to the crushing mill, and all large pieces broken in two or more pieces by sledging to avoid as far as possible the risk of getting lead into the crusher.

But in spite of these precautions the machinery was frequently stalled, belts thrown off, and both belts and bolts broken, and finally the frame of a 9 x 15 Blake's crusher was split down through the jaws. The shaft was also badly bent and had to be straightened and rebabbitted in the bearing where the great strain had squeezed out the babbitt. But by this time rapid strides were being made in the consumption of the speiss pile that had been an eyesore for so many years, and after being encouraged by a 5 per cent. silver gain in one month's run over and above the silver charged in the smelting returns, it was seen that the heavy repairs to several crushers could be afforded.

There was no attempt to keep the records of the copper furnaces separate from the lead smelting, since the copper smelting was carried on with a view of desilverizing dirty lead slags in combination with straight matting for direct profit,

so that the 5 per cent. gain represented a total of 15,000 ounces on a charge of 300,000 ounces, and also no loss on the silver charged, which under favorable conditions would have amounted to about 9000 ounces, so that at a low estimate the gain was about 24,000 ounces. The contents of the speiss were not charged to the furnace, and so long as the smelting returns continued to show in that unusual and satisfactory fashion it was deemed advisable to go on utilizing it.

The crushing continued day and night; slowly, to be sure, but fast enough to use the stock on hand in about five months —all too soon for the writer, for by that time we had become speiss hungry and the dump was thoroughly explored by tunnels in the hope of finding more buried treasure.

Next in importance to the crushing came the roasting of the crushed product. This was carried on in reverberatory calciners of the standard type, with hearths 14 feet wide by 70 feet long, the speiss being mixed with crushed sulphides of copper and iron in the proportion best suited to driving off the maximum amount of arsenic and sulphur without becoming too fusible. It was found that if the speiss constituted too large a portion of the roaster charge it was likely to become plastic as it came near the firebox, and in that condition would stick to the hearth, as well as to the paddles and rabbles, and would naturally refuse to part with its sulphur and arsenic.

After experimenting with different roaster charges it was finally found best to use one-third speiss and two-thirds ore as giving the most satisfactory results, and this charge was continued until the stock was used up. The ore that was roasted with the speiss contained about 27 per cent. SiO_2, 25 per cent. Fe, 30 per cent. S, 7 per cent. Cu, and was admirably suited to the purpose, as the silica rendered the roaster charges more infusible than they would have been otherwise, and the copper furnished the carrier for extracting the silver in the blast furnace. The roasters were charged four times each shift with 3000 pounds of the mixture.

The charge was worked the same as if it had been all ore, rabbled every twenty minutes and moved forward four times each shift of twelve hours. With five roasters running, each one handling twelve tons per day, we roasted twenty tons of speiss and forty tons of ore down to about 5 per cent. sul-

phur. The arsenic was partially driven off, about one-third the original amount remaining in the roasted product, which I am inclined to believe was oxidized to a considerable extent. The charges were drawn from the furnaces into slag-pots and dumped on to a cooling floor. In order not to alarm the men unnecessarily and at the same time avoid danger of poisoning, the proportion of speiss to ore on the charge was increased gradually from 10 per cent. at first, to 15 per cent., 20 per cent., 30 per cent., and 40 per cent. of the charge. But with so much as 40 per cent. or 50 per cent. the fumes, when drawing the charges, became dangerous, which, with the disadvantage of the plastic condition referred to previously, made it both necessary and desirable to decrease the amount of speiss. Some of the men were badly poisoned about the nostrils, and, in fact, one was dangerously ill from the effects, but on the assurance of the company's physician that this could be guarded against, the work was kept up. A preparation of hydrated oxide of iron was put up as a salve and furnished to the men to put into their nostrils, and whether this preventive was the cause or not, we did not have any further serious trouble or fatalities.

The calciners and the calcined material, after cooling, were covered by a white sublimate of arsenious acid, which had the appearance of frost on a cold morning. After cooling from twelve to twenty-four hours the roasted material was loaded on cars and sent to the blast furnace, where it was smelted with silicious copper ores from the Sedalia mine, Colorado, or from the Eureka Hill in Utah, together with the addition of enough raw sulphides from the Leadville mines to keep the matte production at such a ratio that the furnaces would run as regularly as possible. As before stated, these furnaces were originally lead furnaces, with the crucible filled with brickwork and silicious lining.

The jackets were of the ordinary cast-iron type in use in all lead smelters and of the ordinary height, *i. e.*, about four feet six inches. This height of jacket does very well for a lead furnace where the blast is not high enough to raise the zone of fusion, but in the smelting of these ores there was great trouble from burning out of the brickwork over the jackets. The furnaces would treat about 40 tons of ore per day on a 16 per

cent. fuel consumption, and, in addition, 20 tons of slag. It would naturally be expected that smelting with so much speiss on the charge a considerable quantity of speiss would be produced and would separate from the resulting matte. But such was not the case. When the furnaces were tapped it would frequently spark in the way which is characteristic of speiss, but after cooling there would be no line of separation in the pots, and upon being crushed and roasted and resmelted the product was a matte of very clean appearance with 40 to 50 per cent. copper, the arsenic contents of which did not exceed 5 per cent.

CHAPTER II.

Extraction of Gold and Silver from Matte.

The matte produced at the Arkansas Valley Smelting Company's works was shipped to the refinery at Argentine, Kansas, and treated by the improved Hunt & Douglas process, or reshipped to Block & Hartman, Belleville, Ill., and there treated by a leaching process.

In the Hunt & Douglas process the matte is roasted at very low temperature, so that copper sulphate and oxide result without forming any silver sulphate. It is then leached with dilute sulphuric acid, the gold, silver, and lead remaining in the residue. The copper solution is chloridized by the addition of chloride of lime and the copper precipitated as subchloride by passing sulphurous acid through the solution. The subchloride of copper is reduced to suboxide by milk of lime, whereby chloride of calcium for further use is recovered, while the suboxide of copper has only to be reduced to ingot by a simple smelting.

The Block & Hartman process for recovery of gold is somewhat similar to the practice at Argo, Colo., if not identical with it. The matte is first roasted to convert the silver to sulphate, when it is leached out with water and precipitated on metallic copper, the gold remaining behind in the matte. Or it is concentrated to black copper in the reverberatory furnaces, granulated, ground, roasted, and leached with salt solution by the Augustin process and the silver precipitated in the usual manner.

The residue containing the copper and gold is then concentrated in a reverberatory furnace until a small amount of copper is extracted as a copper bottom, carrying nearly all the gold. The subsequent treatment and separation of the

gold from these copper bottoms has been much talked of among metallurgists, and is supposed to be a secret of such importance that Mr. Pearce, of Argo, though generally most liberal in imparting knowledge, is for business reasons unable to divulge it.

An experience that occurred while running the furnaces at Leadville bears on this point, and the reader may be left to form his own conclusions about the separation of gold from copper bottoms at Argo. In concentrating the matte which, owing to the large percentage of lead slag on the charge, contained a small amount of lead, there were formed some bottoms which, while they could not be called properly copper bottoms, will answer for an illustration. These bottoms were metallic in appearance and would ring like bell metal when struck with a hammer, were quite malleable, and could not be broken. The matte, which separated from them very easily, did not contain more than 55 per cent. Cu, so that without the intermixture of lead they would not have been formed. Having them on hand it became a question what to do with them. An assay showed that the gold had almost all left the matte and gone into these bottoms. The experiment of fusing a portion in a scorifier with the addition of a pyritous ore containing no silver and a little gold was then tried. It was found that the gold continued to concentrate in the portion which remained metallic, while very little, if any, went into the matte formed by the pyrites on the surface of the fused charge.

In fact, there was a scorifying action, with sulphur as the agent of concentration, and the resulting metallic portion all the time growing richer in gold as it got smaller. This action was kept up until the greater portion of the copper had been removed, and the resulting button could be easily cupelled with lead whereby the remaining copper was removed, and a gold button remained which could be dissolved and precipitated as fine gold. Certainly the gold can be extracted from copper bottoms in this way, and since it is a well-known fact that mill concentrates and even tailings are to be had from certain gold mines around Black Hawk, Colo., with which the excess copper could be reconverted into matte and the gold concentrated into such, a small amount of copper as to admit of its being refined

it remains for the reader to decide whether it is a secret process or only one of the tricks of the trade. It is at all events only applicable to just such conditions as exist in this process of extracting the silver first and subsequently the gold, and it is difficult for anyone to see how a competitor could possibly take advantage of the general knowledge of the process, especially as the electrolytic method is better and cheaper.

CHAPTER III.

The Calculation of Furnace Charges.

The calculation of a slag for the furnace is illustrated by the following approximate make-up of the charges we were using in matte smelting at the Arkansas Valley works, as described in the previous chapter, taking the analysis of the ores from memory. They may not be exact, but they are close enough for the purpose of illustrating the method ordinarily used by metallurgists for figuring blast-furnace charges.

	Weight in lbs.	Per cent. SiO_2.	Pounds SiO_2.	Per cent. FeO.	Pounds FeO.	Per cent. CaO.	Pounds CaO.	Per cent. S.	Pounds S.	Per cent. Cu.	Pounds Cu.
Calcined ore and speiss	500	20	100	42	210	0	0	5	25	5	25
Raw sulphide ore	150	26	39	32	48	1	1.5	30	45	8	12
Silicious ore	150	60	80	14	21	6	9	2	3	3	4
Lime rock	200	3	6	. .	. .	52	104	. .	. .	. .	. .
	1000	. .	225	. .	279	. .	114	. .	73	. .	41

There is no algebraic mystery or X, Y, and Z equation to be solved by higher mathematical methods than percentage, and the results are just as reliable. There is generally a great deal of mystery thrown about this portion of the metallurgist's work, as if there were a fear that if the younger members of the profession were put in possession of the combination they might soon be competitors.

The writer has no hesitation in saying that the calculation of a charge, when it has been decided what ores are to be smelted, is the simplest thing about the work. The difficulty is more often to decide what to put on and how much matte to make to have it run to the best advantage. There are cer-

tain things a metallurgist must know; for instance, he must be able to tell approximately how much matte the charge is going to make and how much of the iron is going into it in order to make the proper deduction from the total amount on the charge. The writer has been accustomed to have all analyses of ores, whether sulphides or oxides, determined or figured as FeO, and then, knowing from experience and from the running of the furnace about the grade of matte that certain charges will produce, he proceeds to multiply the total pounds of copper by a certain factor to get the weight of matte that the charge will produce.

In the case of the charge above cited let us assume that the matte will assay 25 per cent. Cu. He accordingly multiplies the 41 pounds of copper in the charge by four, which gives the matte production at 164 pounds. A 25 per cent. Cu matte with the amount of impurities, such as Pb and As, that would naturally go into it from ores of ordinary character, would contain the equivalent of 40 per cent. FeO, and 40 by 164 equals 65.9 pounds FeO, to be deducted from the total of 279, and leaving off the tenths gives 214 pounds FeO to go into slag. The sum total of the SiO_2, FeO, and CaO, after deducting the iron that will go into the matte, is 553 pounds, which according to slag determinations is generally 90 per cent. of the slag.

Accordingly 553 pounds of SiO_2, FeO, and CaO, when put into slag together with Al_2O_3 and other oxides, would represent approximately 620 pounds slag to a 1000-pound charge. These 620 pounds divided into the amounts of SiO_2, FeO, and CaO that would go into the slag would give a calculated analysis of 36.3 per cent. SiO_2, 34.5 per cent. FeO, and 18.4 per cent. CaO.

In explanation of the reason why it is assumed that the matte produced will assay 25 per cent. Cu, it may be stated that this would be the experience with ordinary running of the furnace on the charge figured, the excess of the sulphur being burned off. The grade of matte being dependent on the condition of the furnace, the depth of charge, and the amount of blast used, a knowledge of the effect of these conditions is necessary in order to make this assumption.

As regards the percentage of Fe in different grades of matte, that is a matter of local experiment depending on the

amount of other impurities that the matte can absorb from the ores on the charge. For example, if the charge contained no arsenic, antimony, or zinc, the iron contents of a 25 per cent. copper matte resulting from smelting them would be higher than if these impurities were present in the ores. If present they will go partly into the matte and will displace iron.

Roughly speaking, matte is a compound of the sulphides, arsenides, and antimonides of the metals, while slag is a union of the oxides of the metals with the oxide of silicon. The characteristics of either matte or slag as regards melting point, conductivity, and specific gravity are capable of as many variations as the composition. As in the case with the alloys of the metals themselves, where by certain combinations it is possible to produce one that will melt at a much lower temperature than any of its constituents, so it is with slags and mattes, but in a less marked degree. There are certain combinations of SiO_2, FeO, and CaO that form the more fusible slags, and are adapted for certain purposes.

The science of metallurgy is the application of this knowledge to the formation of fusible compounds in order to assist in the recovery of the metals or their sulphides, arsenides, or antimonides. In the case of the charge figured above the slag would be silicious enough to insure the driving off of a reasonable amount of sulphur and arsenic and still near enough to a neutral slag to flow freely, and in view of the grade of matte produced should be clean or at least as low as 1 per cent. of the matte assay.

The assay of slags is a thing as much dependent upon the means provided for settling them and the care with which they are handled as upon their composition. Still it bears a very close relation to the assays of the matte or bullion, which are closely related to each other, subject to varying conditions. Generally speaking, the slag will assay 1 per cent. of the matte in copper work and 2 per cent. in lead work. In the latter case the matte will, as a rule, contain about one-fifth as much silver per ton as the bullion if the amount of matte is normal; but if the matte production is large compared to lead, the relation between assays will fall lower. In a case where 400-ounce bullion was being pro-

duced on a 11 per cent. lead charge with 5 per cent. of matte, the matte would assay approximately 80 ounces, and the slags if well settled, 1.6 to 2 ounces. But these things are so variable that hardly any rules can be given that will not have as many exceptions as applications.

The calculation of a charge fcr the copper furnaces at Aguas Calientes, Mexico, is here given:

	Weight in lbs.	Per cent. SiO_2.	Pounds SiO_2.	Per cent. FeO.	Pounds FeO.	Per cent. CaO.	Pounds CaO.	Per cent. S.	Pounds S.	Per cent. Cu.	Pounds Cu.
General mixture	500	25	100	30	150	7	35	20	100	6	30
Mixture	200	20	40	36	72	2	4	33	66	0	0
Silicious ore	200	50	100	7	14	3	6	4	8	0	0
Copper ore	100	20	20	19	19	4	4	22	22	19	19
Converter slag	150	25	37	62	93	0	0	0	0	5	7
Lime rock	260	3	8	. .	. . .	52	135	. .	. . .	. .	. .
Concentrates	90	19	17	32	29	2	2	21	18	10	9
	1500	. .	322	. .	377	. .	186	. .	214	. .	65

Assuming a 25 per cent. Cu matte, 65 pounds Cu $\times 4 = 260$ pounds matte $\times 40$ per cent. $= 104$ pounds FeO to be deducted for matte, leaving 273 FeO to go into the slag. 322 SiO_2, + 273 FeO, + 186 CaO $= 781 \div 9 = 870$ pounds slag to the charge. Dividing 870 into 322 $SiO_2 = 37$ per cent.; into 273 FeO $= 31.3$ per cent.; into 183 CaO $= 21.4$ per cent.

In the figuring of charges the estimation of the SiO_2, FeO, and CaO contents to the one-tenth of pounds is a fineness of calculation that is totally wasted on the man with the shovel at the charge scale. Besides, many other conditions prevail which do not warrant the assumption that the ores are uniformly of the same composition. The bedding may not be regular, and when a new bed is opened more of the top layers go into the charges than later, and if the bed is in a square bin, as the opposite side is approached, the top layers are exhausted and the bottom layers alone will for a time take the place of the entire bed. In the face of such conditions calculations that may have been perfect melt into nothingness, and it becomes a necessity to be able to tell the character of the slag as it comes from the furnace and make such corrections as are necessary, frequently without analysis.

The larger the ore mixtures the more evenly the furnaces will run, but if the mixtures are small and it is necessary to change the charge every day, there is not sufficient time to settle down to business and get the charge corrected to just the proper point to do the best work. The weighing of the charges is a part of the operation that requires the constant attention of a careful foreman. Owing to the scattering of fine ore over the scales while shoveling in from the bins to the car or barrow, the scales should be swept clean after each charge and the accuracy of the balance frequently verified.

The character of slag to be run on is largely a matter of personal preference with different metallurgists, but in general it is determined by the local conditions regulating the cost of iron and lime.

The limits within which the experience of the writer has reached the best results are for copper blast-furnace work:

SiO_2, 30 to 38 per cent.; FeO, 30 to 40 per cent.;
CaO, 10 to 25 per cent.

But under ordinary conditions the slag that is best adapted to a constant discharge and regular run is:

36 per cent. SiO_2, 33 per cent. FeO, 21 per cent. CaO.

It may change considerably either way and continue to run just as well, and a slight correction will bring it back to the original type.

I do not pretend to say that slags can not be made higher in SiO_2 than 38 per cent., but they run very slowly compared to slags with less, and the coke seems to burn out faster than the charge smelts, leaving the furnace full of cold stuff, and accretions form rapidly at the tuyeres where the blast strikes the slag.

It has been stated in other works that the composition of the slag may vary without serious results within much wider limits than those stated by the writer. While this is true of furnaces where the slag is tapped periodically and can collect in the furnace and run out with a rush, it is not true of a constant-flow furnace, as any such slags would solidify in the spout, and before the furnace could be opened the accumulation in the furnace would run out from the tuyeres. At

Anaconda, where the charge was largely slag from the copper converters, I have often had the analysis show 40 SiO_2, 43 FeO, 9 CaO when the matte production was large (about 30 per cent. of charge), but in spite of this the furnace would run much slower and the spout would be difficult to manage. It was found that when the SiO_2 was near 36 per cent. and the sum of the FeO and CaO about 54 per cent., the best results were obtained. This was also the experience with the copper furnaces at Aguas Calientes, and while it might appear to be a better commercial proposition to make a more silicious slag, still, if the furnaces will not smelt enough ore on that kind of slag to supply the material to keep up the circulation in the spout and prevent its freezing, it does not pay to run on such slags.

CHAPTER IV.

Types of Furnaces.

The type of furnace has much to do with the kind of slags that can be run in it. An intermittent tap will handle slags that would not do at all for a constant flow, for the reason that in the latter case the slag would chill so quickly that the taphole would be choked up, causing shutdowns that in many cases would develop into freeze-ups. The best type of intermittent tap is a single slag-tap in front and matte-tap on the side at about one foot lower level. The hearth should incline forward from a foot below the level of the tuyeres at the back to two feet in front. This is a better arrangement than having the matte-tap immediately below the slag-tap in front, for the reason that the matte can be tapped at any time while the slag is running without removing the settler. The constant-flow furnace, on the other hand, gives better results when the matte and slag are run into a forehearth of from 5 to 10 feet in diameter and are given ample time to separate. The volume held in ordinary overflow pots is too small for complete separation. Like the deposition of sediment from water, matte and slag must come to a standstill in order to separate thoroughly.

A forehearth requires considerably more tonnage from the furnace to keep it open than the shallow crucible and overflow pot, and also as the settler is made larger more matte on the charge; otherwise the radiation is enough to form a very thick crust of slag on the top and sides and greatly diminish the size of the settler. The interior of the settler takes the shape of a pear with the larger end down, where the corrosive action of the matte is the greatest, while in the upper portion the slag once chilled remains there.

The handling of furnaces with internal crucible is attended with about the same amount of difficulty as those with forehearths, the difference being that in the first case, if it is impossible to tap the matte from the furnace, it can be run out into the settler in front until such times as the accumulation in the furnace renders the breast soft enough to drive a bar. But if the matte-tap on the forehearth is lost through failure to put in a bar immediately after tapping matte and while it is yet soft, or through chilling of the contents owing to slow running of the furnace, then all is lost; for while the furnace may be put into good condition in a few hours by matte charges, the settler being lost and too large to move, there is nothing to do but blow out the furnace and dig out the settler. This is the only bad feature about large settlers; every other point is in their favor, especially in connection with converters, as they will permit a sufficient amount of matte to accumulate for the converter charge.

The separation between matte and slag is much better in large than in small settlers, but the weight of a settler 10 feet in diameter by 4½ feet deep, filled with slag and matte, is too great to move it unless extraordinary appliances are provided. In all cases after tapping matte, unless the breast is exceedingly soft, as soon as the flow has been stopped with clay, a steel bar should be driven in slowly until the point has just penetrated the clay and entered the crucible or hearth. This is to provide against a hard breast when the next tap is to be made, and is a very important part to attend to in the running of the furnace. If, for instance, it is neglected, as it is likely to be owing to the attention of the men being drawn to other things, it may mean several hours of sledging to again open it although the breast be reasonably thin.

With the matte-tap immediately below the slag-spout, as was the case in Leadville, the slag was drawn off until the blast blew out at the taphole. The settler was then removed as quickly as possible and the bar driven into the matte-tap, a slag-pot run under the matte-spout and the bar withdrawn by means of the ring and wedges, which are absolutely necessary. All the matte in the furnace would be tapped out without stopping the flow, except for a few seconds to change pots. The stoppage was done by means of a pole consisting of a

piece of iron shaped like a tinner's soldering-iron, about 15 inches long and 2½ inches in diameter, welded to a gas-pipe handle about 10 feet long. The heavy point would be inserted in the taphole and the stream interrupted until the full pot could be drawn away and an empty one substituted.

The taphole in front is not a good arrangement. It should be on the side of the furnace but as near the front as possible. Its position in front at Leadville was unavoidable on account of the way in which the furnaces were crowded together. It frequently happened that after the settler had been removed and before the matte-tap could be opened, the furnace would be full of slag and the slag-tap have to be opened again to prevent it running out at the tuyeres, which it sometimes did. With the matte-tap on the side the work of tapping matte can go on at any time without interfering with the slag-tap or necessitating the removal of the settler. This point is worth mentioning, because if it is known how not to do a thing it may be of great assistance in finding out how to do it.

At San Luis Potosi, Mexico, they have a number of lead furnaces converted into matting furnaces by filling the crucible and putting in a sloping hearth of firebrick. The crucible is about two feet deep below the tuyeres in front, and a tap-jacket is put in on the side near the corner of the furnace, and high enough above the floor to admit a slag-pot under it, with slightly depressed floor space immediately around the tap. These furnaces are used for the concentration of matte made in the lead furnaces, whatever metallic lead there is produced coming out with the matte and being recovered on the dump.

Running a blast furnace in connection with a converter plant of the size of the one at Anaconda, where the sole purpose is to work up refuse material and smelt over converter slag, is a much simpler operation than smelting ores which are widely different in character and only mixed mechanically in the charge. The fact that a portion of the charge has been fused previously, no matter what the composition of the fused material may be, has a very beneficial effect on the smelting. This is probably due to the fact that the fused material melts higher up in the shaft than the unfused and has a dissolving and uniting effect on the constituents of the latter.

With a charge that was 60 per cent. converter slag, it was

only necessary to add raw ore to make the matte sufficiently low to extract the copper from the slag and waste material, together with lime rock to assist in the fluxing of the surplus SiO_2. The FeO was already in the converter slag and in combination with SiO_2, so that such smelting can not be said to require any special mention from a scientific point of view, but merely as a matter of practice in connection with converting to recover the copper in the slag

CHAPTER V.

Spouts, Settlers, and Jackets.

The form of spout and tap-jacket for a constant-flow furnace is a very important matter and one of which no special mention has been made in any work on the subject. The spout ordinarily in use in Montana is made of one-inch pipe put together as closely as possible, and in such a way as to form a channel of four and one-half feet long, semicylindrical in shape, with one end open and one partially closed. A piece of sheet iron is fastened to the curved outside, and clay or a mixture of ground quartz and clay is rammed in between the pipe and against the sheet iron. When in position with the open end against the front of the furnace (see drawing of Anaconda furnace) the spout allows the slag to flow in a steady stream into the forehearth without the escape of blast, the slag stream flowing over the partially closed end and thereby trapping the blast. This spout is known as the Schumacher spout, and has been in use for many years. It does good work, but the weak point about it is that the quartz or clay is eaten out by the slag and matte, and frequently the contents of the furnace flow through some defective point in the spout.

To overcome these defects and prevent runaways the writer had the coil cast into an iron spout of the shape shown in Figs. 5 and 6. It was found that fewer pipes were necessary for this kind of a spout than with the old construction, and the number of coils was reduced to four.

The cast iron proved to be sufficiently cooled by the water circulating through the coils to keep the spout from being attacked by the matte, except at the tip, where the stream falls into the settler. At this point the cast-iron covering is eaten away and the pipe is soon laid bare.

If the pipe used in making the spout is not of the best quality it may soon be eaten away at this point and the spout rendered worthless. This is the most satisfactory spout that the writer has ever used on a constant-discharge furnace, as the cooling effect of the water is reduced to a minimum and is only what is required to keep the spout from being eaten out by the matte.

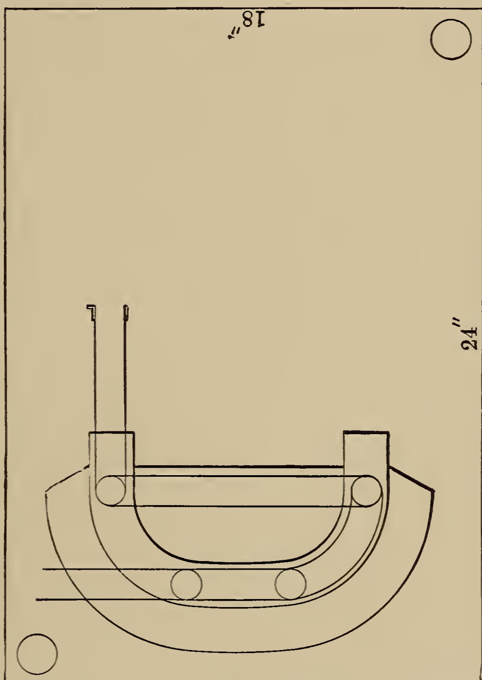


FIG. 5.—THE HIXON SLAG-SPOUT; CROSS-SECTION.

The pipe used may be of any size, say from $\frac{3}{4}$ to $1\frac{1}{4}$ inches, according to the water pressure, but must be of the best quality and of extra thickness. The coil should be made very carefully and put together with malleable ells and return bands. It should then be heated to drive off all grease or oils, and while warm painted with graphite mixed in benzine. Several coats of this should be put on and allowed to dry before the coil is put into the sand mould to receive the cast-iron covering. The cast should be poured as cold as possible in order to avoid burning the pipe.

Any good foundryman should be able to make these spouts without much difficulty, but still it sometimes happens that the pipe will be plugged by being fused and great care is necessary.

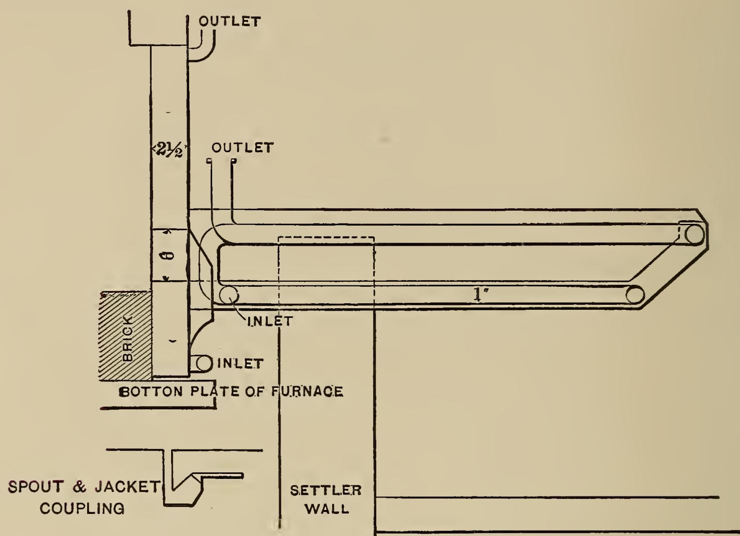


FIG. 6.—THE HIXON SLAG-SPOUT; LONGITUDINAL SECTION.

It is best to have the tap-jacket made of bronze, but I see no reason why a good quality of cast iron, or especially malleable iron, would not do, although it would be more liable to crack. There is a yoke cast on the front of the jacket immediately surrounding the taphole which is cored out for water space. The inside dimensions of this yoke are the same as the outside dimensions of the spout, and the yoke is to prevent the escape of slag and matte from between the spout and tap-jacket. The yoke should be cast larger near the jacket and the spout dovetailed, so that when the two are put together they will fit tightly and need no bolts or other clamps to keep them from springing apart.

Spouts have been made in other ways, some of boiler iron, flanged and riveted together in such a way as to form one of much the same shape as that described, but the trouble is that there is too much cooling effect on the stream of slag flowing from the furnace, the result being that a skull forms in the spout which grows thicker rapidly, and finally the

stream is interrupted entirely by the closing up of the channel. The furnace goes on smelting and the slag soon runs out of the tuyeres because it has no other outlet from the furnace. A spout with the coils, such as that described, exerts the least possible cooling effect on the slag stream, and if the furnace runs at any reasonable tonnage and the slag is within the limits of composition mentioned on a previous page, will keep open and give no trouble at all from freezing up. A fire, either of wood or lump coal, should be kept on top of the slag stream, and by this means the stream should be liquid and show no shell or crust from the furnace to the end of the spout. Spouts constructed of boiler iron are a constant source of annoyance and will freeze up when the furnace is running at its best.

At Anaconda the settler was much smaller than at Aguas Calientes, and of a different type. (See Fig. 7.) The slag and matte were discharged together from the spout into the settler, where the matte by its greater gravity would go to the bottom and pass under the partition of pipe coil and discharge at a lower level than the slag at the opposite end. This partition and the lining of the slag end of the settler were originally made of brick, but it was found that when the furnace ran a heavy tonnage the partition would be destroyed as well as the lining in the slag end of the settler. Pipes with a water circulation were substituted and worked very well as long as the tonnage was kept up, but would chill the contents of the settler if it became low, or was shut down for more than twenty minutes without tapping out. It is interesting to note that the copper contents of the slag discharged from these small settlers at Anaconda were much higher than slags made at Aguas Calientes, where the settlers (Fig. 8) were the largest in use (10 feet in diameter). On a charge producing 55 per cent. Cu matte at Anaconda the Cu contents of slag would average nine-tenths per cent., while under the same conditions at Aguas Calientes the Cu would not be more than five-tenths per cent., showing that the capacity of the settler has much to do with the cleanness of the slag. With lower grades of matte the slag assays would be lower, but about the same relation existed in the two cases.

The jackets for a copper furnace of large size can be arranged in many ways, though there are but few good designs. In

some cases the furnace is all one jacket from the crucible or base-plate to the feed-door, and the other extreme was embodied in the furnaces at Aguas Calientes, where there were twenty-four jackets on the furnace, counting the tap-jacket and spout. It is a very poor plan to multiply troubles unnecessarily in any business, and especially water-jackets.

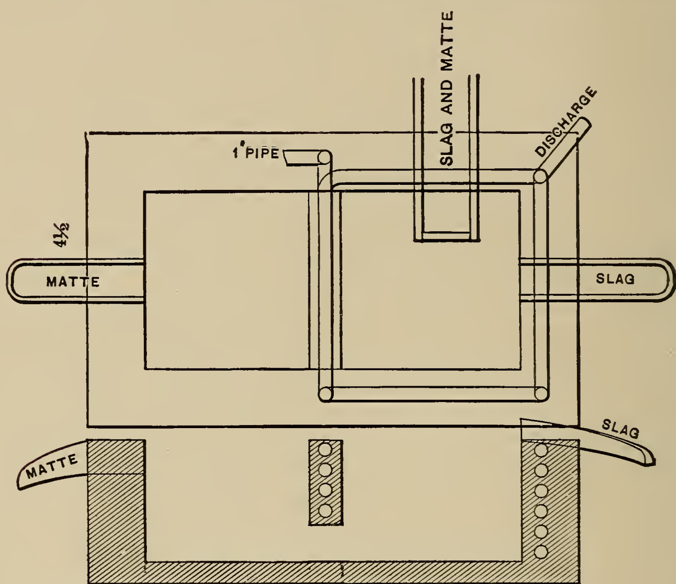


FIG. 7.—PLAN AND SECTION OF SETTLER USED AT ANACONDA.

The greater the number the more opportunities there are for one to leak or the water connection to become clogged and allow the jacket to burn out. On the other hand, it is not advisable to make the jackets too large. The middle course is the best. The front and back should be of two jackets each (see Fig. 9). The sides may be divided into any number to suit the conditions, but so that there shall be no seams or rivets exposed on the inside of the furnace. Jackets become very expensive as the size increases, on account of the unusual width of sheets necessary for the inside. Better results can be obtained by dividing the side of a 120-inch furnace into three or four jackets of equal dimensions. With three jackets to the side and two tuyeres to the jacket, there are six tuyeres to the side and twelve tuyeres to the furnace, which

is quite sufficient, providing they are made 4 to $4\frac{1}{2}$ inches in diameter. Four jackets to the side and two tuyeres to the jacket gives 16 tuyeres to the furnace, which is rather too many, but if they are made 3 to $3\frac{1}{2}$ inches in diameter they

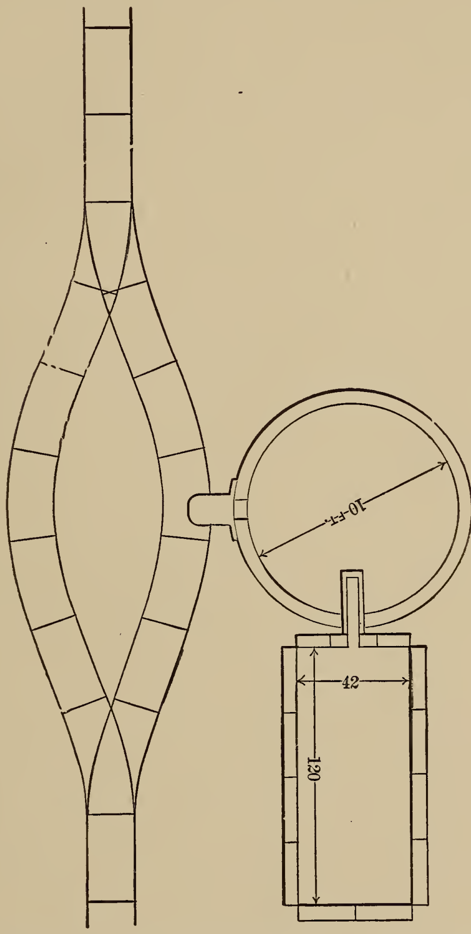


FIG. 8.—PLAN OF FURNACE AND SETTLER AT AGUAS CALIENTES.

will be quite satisfactory. The objection to a large number of tuyeres is that they will be so close together that the noses that form in front have a tendency to unite and form a dark band all the way around the furnace, causing the zone of fusion to travel up. It is a mistake to have the tuyeres point

downward, as is seen in some furnace drawings. The reason is that in case slag fills the tuyeres, as it is sure to do sometime in a long run, the slag can not escape and solidifies in the tuyeres, whereas with a simple conical-shaped tuyere put in with its axis perpendicular to the face of the jacket, the slag can escape until the tuyere can be plugged and the trouble rectified.

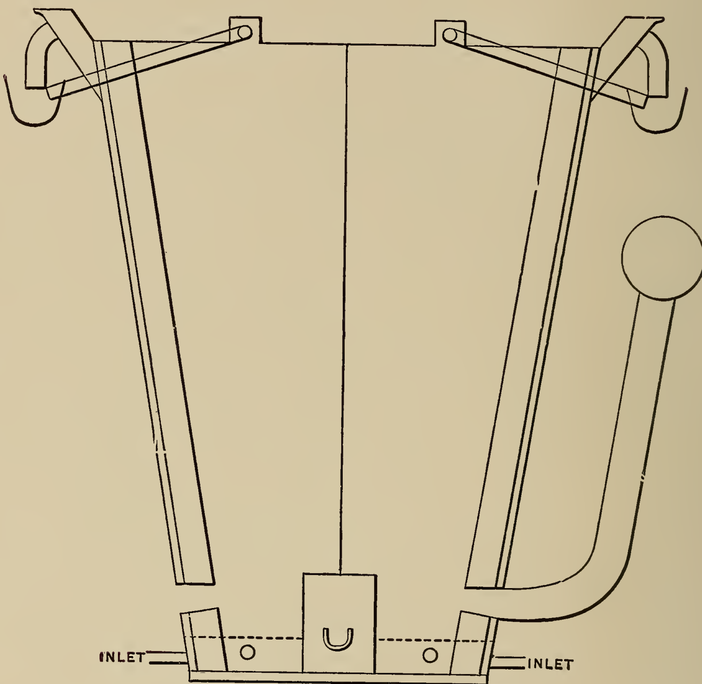


FIG. 9.—DESIGN OF WATER-JACKETS.

It is also a mistake to put the jackets in a double row, one above the other, for the reason that whatever mud or scale there is in the water will settle in the bottoms of the upper row, and that point being above the tuyeres where the heat is greatest will burn out and cause great trouble and unnecessary expense, to say nothing of filling the furnace with water and freezing it up. The writer has had experience with furnaces built in this way and is thoroughly convinced that no greater mistake can be made than putting one set of jackets above another.

The height necessary for the jackets to extend above the tuyeres is dependent upon how the furnace is to be run. If it is to be run on high blast and crowded to its greatest capacity, then the jackets should extend to the top of the charge, say 10 feet above the tuyere centers, but if there is to be low blast and easy running, as in lead smelting, then they do not need to be so high, since a shaft of firebrick will serve quite as well. Generally speaking, the jackets for a copper furnace should extend from 8 to 10 feet above tuyere centers, which should be from 18 to 24 inches above the bottom, making the jackets from 9½ to 12 feet in total height, and of a width depending on the number used and the size of the furnace.

The jackets for the Anaconda furnaces were made of about the same height as those for lead furnaces. This was found to be a mistake, as the burning out of the shaft proved later, and to correct it a series of coils of 2½-inch pipe were put in as a lining to the shaft of the furnace from the top of the jackets as far up as the brick burned out. The pipes were first put together with cast ells, but these broke and malleable ells had to be substituted.

There were three pipes to the foot and about six feet of the shaft were lined, so that it required about 500 feet of pipe and 54 malleable ells. The water was brought in at the bottom coil and made the circuit of the furnace eighteen times before escaping. This method of protecting the shaft is much cheaper than high jackets, and in case the water does not scale or contain much mud, will give equally good results. The method is mentioned here for the purpose of showing a way of changing a lead furnace to a copper furnace.

Under stress of circumstances, and in a locality whither it would be difficult to transport large jackets, a blast furnace could be easily and cheaply constructed with a jacket of pipe 2½ to 4 inches in diameter on the same lines as the brickwork was protected at Anaconda. Starting from the crucible or base-plate the coil should be put in place, each turn slightly larger than the one below to give the shaft the desired taper of about one inch to the foot. When the level of the tuyeres is reached a nipple of 4 inches in length should be put in and the next coil raised by that amount, leaving a space 4 inches

wide all around the furnace, which should be bricked up at all points except where the tuyeres are to enter. Above the tuyeres the coils should be extended to the desired height and clamped securely together at the corners by means of stirrup clamps around the pipe, extending through an iron bar and secured by nuts on the clamps.

The number of coils, malleable ells, and lengths of pipe to be used depend on the size of the furnace, the diameter of the pipe, and the height the jacket is to be carried. For a furnace 40 by 100 inches and a 10-foot jacket of 4-inch pipe, it would require approximately 600 feet of pipe and 100 ells. This would make a jacket that would do just as good work as the more expensive steel jackets of large size and great weight. The water should be brought in at the bottom and forced through the entire coil under a pressure of not less than 25 pounds. If this pressure is not obtainable then the coil should be made in two sections, one above the other and supplied from a 6-inch main. The jacket made in this way will not be open to the same objection as if one steel jacket were placed above another, on account of the flushing out of any sediment by the rapidly flowing water.

Many kinds of fixed tuyeres have been introduced of late years to replace the tuyere bags, but they are not an improvement and are open to many objections to which the simple canvas tuyere bag is not. If, for instance, a jacket is to be removed they are difficult to manage, and, on the other hand, a tuyere bag can simply be twisted and turned out of the way. In case slag comes down from above and fills a stationary tuyere, it is difficult to remove and frequently the tuyere is broken in cleaning. But if bags are used the tuyere can be removed and the opening plugged temporarily with clay. Good heavy duck bags, well soaked in mineral paint, with tuyere points of cast iron, are preferable to all the patent discharge, slag-catching devices that have been introduced.

CHAPTER VI.

Blowing-in and Barring-Down a Furnace.

The method of blowing-in a lead furnace, commonly in use, is to fill the crucible with molten lead by first firing the furnace with wood for several hours to insure its being hot, and then throwing lead on top of the wood fire. A blast of air is blown down through the lead well or from a tuyere in front or back of the furnace to force the fire. By this means 250 bars of lead, or about the amount required to fill the average-size crucible, can be smelted in ten hours. This is generally done on the nightshift, and the furnace cleaned of wood ashes by 7 A.M. A layer of fine wood or charcoal is then put on top of the lead to a height of a foot above the tuyeres, a fire started and the tap-jacket put in place. The tuyeres are left open to admit air to the fuel, and after it has been thoroughly lighted and fire shows in all the tuyeres the furnace is filled up to the top of the jackets with coke. Slag charges are then put on alternating with about 12 per cent. fuel, and after four or more slag charges alone then one slag charge to one ore charge, or two slag to one ore, according to the conditions of charges, etc., until the furnace is filled up to about 10 feet above the tuyeres. A light blast is started and gradually increased to about one-half the regular amount when slag is first tapped. From this time the blast is gradually increased for about four to six hours, when the furnace should be running on full blast, usually at a pressure of about 16 inches of water, but this pressure is dependent upon the fineness of the charge and has to be varied according to the conditions governing the special case.

The barring-down of lead furnaces to prevent the formation of blowholes and consequent high lead loss, is made necessary

by the increase of sulphide and zinky ores treated within recent years. In case the furnace is of the stack type with side feed-doors this can be done by the regular furnace crew in about one shift more or less, depending on the condition of the furnace and the energy and ability of the men. The furnace should be run down by the nightshift so that the breast-jacket can be removed the first thing in the morning. The crust is then broken in and the charge raked out and removed. Two sets of men then start to cut out the shaft from above, while another set work below removing the crusts as they drop down the shaft.

After cutting down the wall accretions from above and removing all loose material from the furnace, a large hole is cut in the crust over the lead well and the furnace is ready to resume operations again with as good results as if just blown in. The wall accretions, if allowed to increase in thickness, will force the passage of the blast through a constantly decreasing space, resulting in greater losses both by flue-dust and volatilization. In case the furnace is of the top-feed type the work of barring down is much greater on account of the greater distance the workmen are from the crusts to be removed, and in many cases it is found better to blow out such furnaces than to bar down, for the reason that while the crusts can be removed in the shaft as far down as the top of the jackets, to work below there with bars 20 to 22 feet long is exceedingly slow and difficult.

Barring-down in some cases is only partial and is preceded by charging the furnace with coke to serve as a bed for the barrings to fall on. After barring in this way, which is generally practiced in the top-feed furnaces, the furnace is filled up with charges and operations are resumed. But it is not by any means so satisfactory as if carried on until the shaft is entirely clean down to the tuyeres, as is the case with side-feed furnaces, and is generally not done more than once in a campaign. In case the furnace is barred down completely, as first described, it is started in much the same way as if it were to be blown in afresh. A wood fire with coke on top to the depth of two feet is made, then from five to ten bars of lead are added, according to requirements, to fill up the hole cut in the crucible, next a few slag charges and then the regular

charge to the depth of about six feet, when light blast is turned on and gradually increased as the furnace is filled up.

To run a furnace down in the best way, either for the purpose of blowing-out or barring-down, it is necessary to reduce the blast, discontinue charges, and have on hand a considerable quantity of coke fines thoroughly wet, to be thrown in a little at a time as the fire becomes too hot. In the case of a copper furnace which is running into a forehearth, it should be allowed to discharge into the forehearth until such time as the stream from the spout is becoming too small to continue to run, when a bar should be quickly driven in the front and the furnace tapped on the side and allowed to discharge the remainder of the slag and matte until the blast is taken off. In cases of shutdown for any length of time the forehearth should be tapped and a bar driven in the matte-tap.

In the case of a lead furnace it results in too great loss of lead to run down to the tuyeres, although there may be four or five feet of wet coke fines on top; therefore the blast should be taken off about the time the charge is down to the top of the jackets and the remainder of the charge raked out on the floor.

It frequently happens from various causes that a crust will form over the lead in the crucible and force the lead out with the slag and matte. A sharp lookout has to be kept on the slag-pots at all times to prevent this, and, in cases where it is possible, to remedy it at once. The causes may be low lead charge, small matte production, bad reduction in the furnace, bad slag, water leaking into the furnace, or any one of a number of causes too numerous to mention. The quickest way is to drive a bar through the taphole down into the crucible, but if the condition of the furnace is not changed by removing the cause, it will soon become impossible to keep the hole open by mechanical means. It is very seldom that conditions of charge and furnace are such that it is at no time necessary to drive a bar into the crucible.

It is stated frequently by men who pose as disciples of infallibility, that "you should not drive bars into the crucible." It is also worthy of comment that furnaces run by these same men have used up considerable steel and have developed some very good strikers. While not wishing to

bear with unusual emphasis on this point as a mark of success, or to state that it is necessary for a metallurgist to help his men by doing manual labor, still it is advisable in cases of this kind either to take hold with both hands or leave it entirely to the foremen and men.

The blowing-in of a copper furnace on a matte charge is not attended with as much difficulty as is the case with a lead furnace. It is important in the starting of any kind of a furnace that the water connections should all be looked after before the furnace is started, in order to provide for the increased use of water during the short period of time before the jackets have become coated with a protecting layer of chilled slag. Before blowing-in a matte furnace the hearth as well as the settler should be well heated with a wood fire for several hours, the spouts being thoroughly dried to prevent explosions by contact with matte.

After this it is only necessary to increase the wood fire in the furnace sufficiently to insure a thorough lighting of the coke covering, which should be put on to a depth of about 18 inches above the tuyeres. When the mass of coke is on fire throughout, charges of one-half matte and one-half impure slag should be put on with coke until the furnace is full. The blast is turned on and gradually increased until the full blast is reached. As soon as slag shows at the tuyeres, the bar is removed from the taphole and the slag and matte allowed to flow continuously into the forehearth. It is better to follow this plan of filling up the settler, especially if it be a large one, than to start at once on ore charges, because it insures the entire mass in the settler being in a fluid condition and avoids the possibility of a hard matte-tap. If the matte and slag are not to be had for the purpose of blowing in, a charge should be figured that will produce the greatest quantity of matte together with a slag that shall run as easily as it is possible to make out of the ores to be smelted. In such a case the coke-bed should be somewhat thicker to insure the ores being smelted before arriving at the tuyeres. It is very important in cases where the settler is large that the slag and matte should enter the settler fast enough to float the crust that will form on top of the fluid mass. In order to assist this as much as possible, men should be stationed around the settler with bars long

enough to be able to break the crust in a line next the walls of the settler, thus allowing the crust to float freely as the settler fills. The slag-tap should be closed to force the settler to fill up about six inches above its usual height when running. This is done to raise the slag-crust into its proper position before allowing it to cool.

When it has risen to the proper height the slag-tap should be opened very carefully and only enough slag allowed to escape to keep the crust from rising still higher, until such time as it has chilled to a sufficient thickness to support itself when the fluid mass is tapped from under it. It may require four to five hours and an occasional sprinkling from a hose to allow the crust to become thick enough. Sometime before the settler is full the slag and matte charges should be taken off gradually and ore charges substituted, so that about the time the settler is full the ore charge should be down to the smelting zone.

CHAPTER VII.

Handling Blast-Furnace Slag.

The handling of the slag from blast furnaces is an important part in their management, and there are four methods now in use for lead furnaces and two for copper which have considerable merit. The first to be considered will be the methods in use by the Arkansas Valley Smelting Co. at Leadville, at the Omaha & Grant works in Denver, by the Mexican Metallurgical Co., San Luis Potosi, Mexico, and the Pueblo Smelting & Refining Co. at Pueblo, Colo. At each of these places one of the four methods for lead furnaces is in use and can serve as an example.

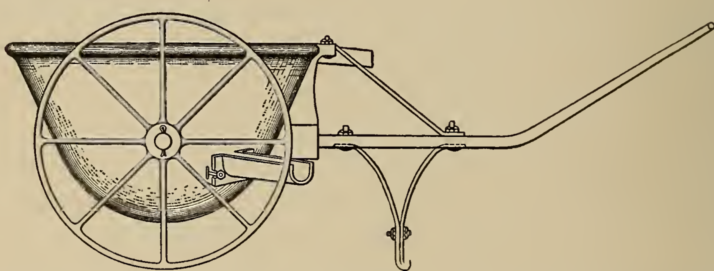


Fig. 10.—Matte-pots used at Arkansas Valley Works, Leadville.

First, at the Arkansas Valley Smelting Company's works the slag and matte are tapped from the furnace into pots of the ordinary or Devereaux type (Fig. 10), and these are run on top of a reverberatory furnace built on a lower level, poured or partly tapped through the Devereaux hole into the furnace, where matte and slag are kept in a fluid condition until a complete separation takes place between them. The slag is then allowed to flow off into larger pots on cars, which are hauled away by an engine, while the matte, when it has accumulated

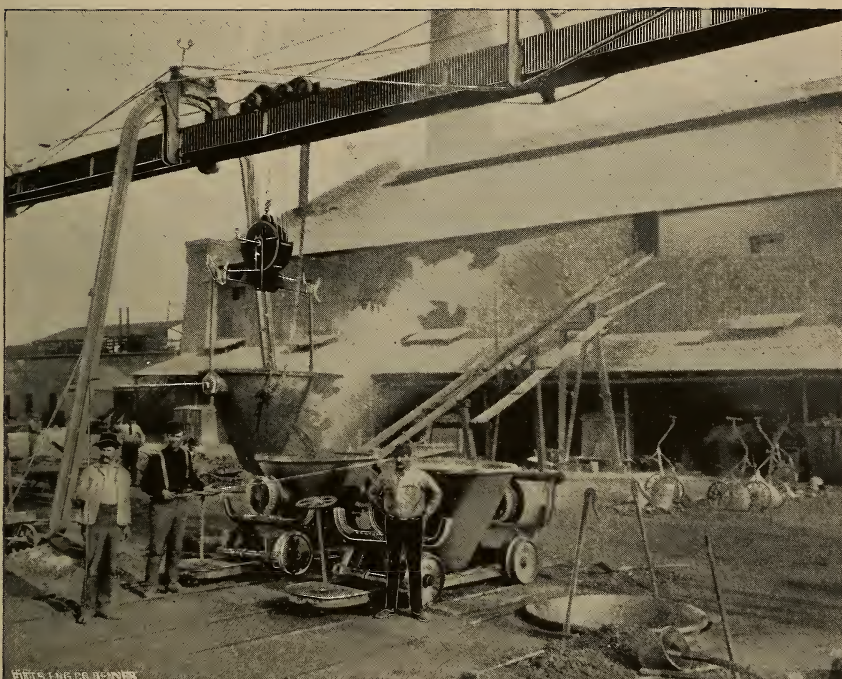


Fig. II.—Matte Settling Pots at Omaha Grant Works, Denver, Colo.

sufficiently, is tapped from the furnace into beds. In this way the settling is an entirely distinct operation from the smelting, and one forehearth serves for several furnaces in blast.

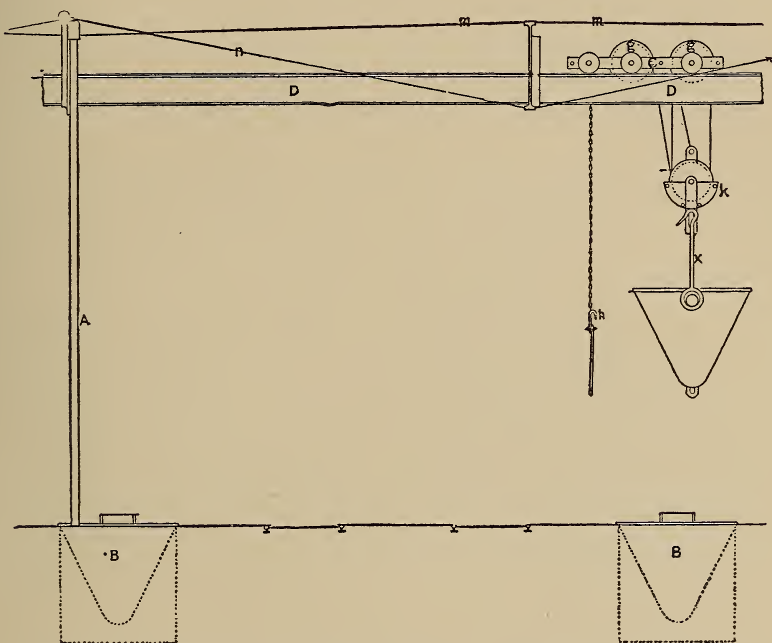


FIG. 12.—MATTE SETTLING POTS USED AT OMAHA & GRANT WORKS, DENVER, COLO.; LONGITUDINAL SECTION.

The second method to be considered is in use at the Omaha & Grant smeltery in Denver (Figs. 11 to 13), where the slag is tapped from Devereaux pots into very large cast-steel pots of the same shape as the Devereaux with the hole about two feet from the bottom, and another hole in the bottom for a matte-tap. These large pots are about five feet in diameter at the top, conical in shape, and about six feet deep. They are handled by a crane and traveler, of very simple but effective design, and can be lifted about and changed with ease. When receiving slag from the Devereaux pots they are sunk in a pit in the dump so that the top of the pot is about an inch above the iron plates which surround it and serve as a floor for the slag-pots from the furnaces. Several small pots can be tapped

into the large one at the same time, and there being two large pots in use one can be drawn off while the other is filling. There is also a dumping pot on a car pulled by a horse to take away the slag after settling the second time (Figs. 14 and 15). The plan of operations is to allow the slag to settle as much as it will in the slag-pots, then to tap out the upper portion into the large settling-pots and allow it to

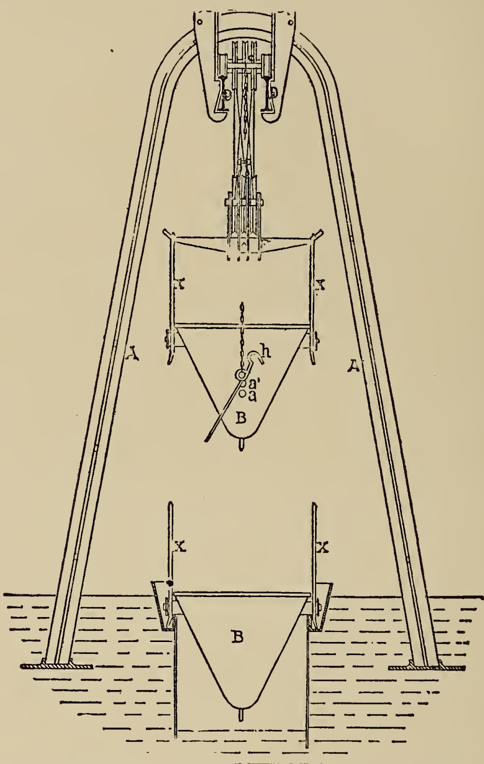


FIG. 13.—MATTE SETTLING POTS USED AT OMAHA & GRANT WORKS, DENVER, COLO.; CROSS-SECTION.

settle again. When the large pot is full of slag it is tapped (as it stands in the pit) into the dump-pot on the car before mentioned, and this slag is hauled away to the face of the dump. If the large settler has not enough matte in it to necessitate a change, it remains where it is and more slag is tapped into it after closing the Devereaux hole, and by repeatedly filling and emptying it of slag the matte that

escapes from the small slag-pots accumulates until it is necessary to change the large settler. It is then hoisted and carried by the overhead traveler to a place provided, where it can be tapped from the bottom and the matte allowed to escape into smaller pots or beds. Meanwhile another large settler is put in its place and the process goes on. The object of this second settling is to collect and recover that portion of the matte which at times will unavoidably escape from the tapping of the Devereaux pots.

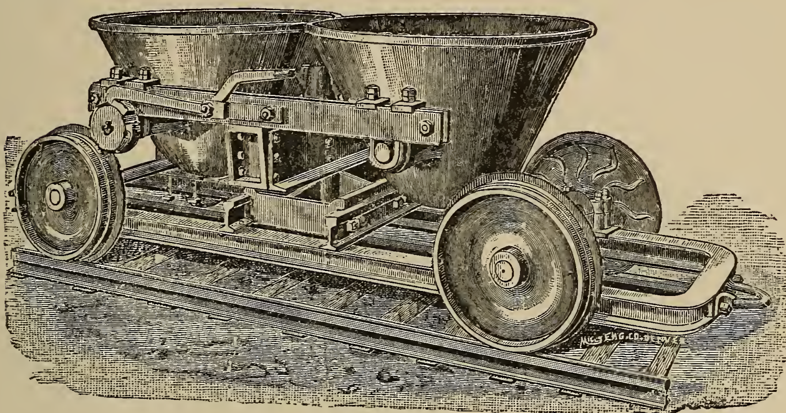


FIG. 14.—SLAG-TRUCK USED AT OMAHA & GRANT WORKS, DENVER, COLO.

The third plan, and the one practiced at San Luis Potosi Mexico, by the Mexican Metallurgical Co., necessitates the use of overflow pots at the furnace, of a size sufficient to collect the matte for from three to four hours, the overflow slag going into the Devereaux pots which are again tapped into other slag-pots of the same size, the contents of the latter being thrown over the dump. The overflow pots are emptied into cast-iron moulds arranged in a circle so that a jib crane can be used to lift out the matte cakes when cool.

The fourth method is in use at the Pueblo Smelting & Refining Co.'s works at Pueblo, and also at the East Helena works at Helena, Mont., which makes the lead furnace a constant discharge, and the matte is tapped at a lower level than the slag, thus making a settler of the furnace.

At each of these places the visitor is given to understand that the method in use is better than all the others, and it

is a difficult matter to decide. However, each has points in its favor to recommend it, and it is to be supposed that the results are about the same.

In case a copper furnace is to be run with intermittent tap, the slag would be handled by one of the methods described for lead furnaces. With a constant flow and settler, the only two methods in use are, first, by pots, which may be either of the ordinary kind on cars and run on tracks, or, second, by granulating with water. The first method needs no special mention except to state that the slag can be handled more cheaply by pots on cars than by the ordinary slag-trucks.

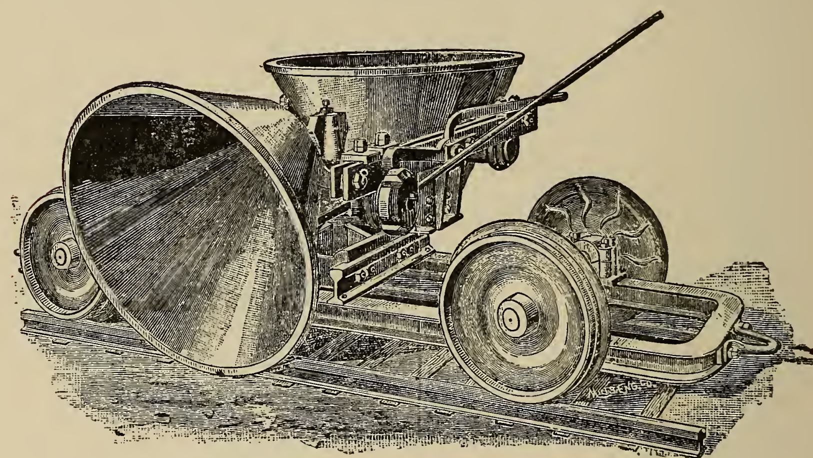


FIG. 15.—SLAG-TRUCK USED AT OMAHA & GRANT WORKS, DENVER, COLO.

The granulating and sluicing away by water necessitates a considerable amount of fall to the sluice, about 5 per cent. being the minimum, and about twice as much water is required as the jackets will use. A six-inch pipe with a pressure of 12 pounds will furnish enough water for the jackets and granulating slag for two 40 by 100-inch furnaces running 125 tons each per day, the water discharged from the jackets being run into the slag sluice.

CHAPTER VIII.

Design of Lead Blast Furnaces.

The tendency of recent years in constructing lead furnaces has been to increase the height from tuyere level to feed floor as well as to increase the area of the furnace.

The dimensions of a lead furnace at the tuyeres have no particular bearing on anything except the capacity, but the height of the furnace above the tuyeres has a very decided bearing on the lead losses, as the experience with the latest type at Aguas Calientes will show. A simple statement of the facts as they occurred will be necessary to show how a lead furnace should *not* be constructed.

The plant at Aguas Calientes originally consisted of a concentrator of about 120 tons capacity, two roasters of a type that no one need desire to imitate, two copper blast furnaces, 42 by 120, with forehearths 10 feet in diameter, and a copper converter 8 feet in diameter by 16 feet high. It was decided to add to this by building two lead furnaces. The copper furnaces were built up on a high pedestal of masonry so that a ladle could be run below the settlers in a position to catch the charge of matte when tapped for the converters. This made the foundation for the lead furnaces considerably lower than the copper furnaces. The charges for both sets of furnaces had to be elevated to the feed floor, and it was important that the charge floors should be on the same level so that in case either elevator was under repair the other could be used for all the furnaces. The lead furnaces were accordingly run up until the charge floor was on a level with that of the copper furnaces. This made them 27 feet high from the furnace floor to the charge floor, or about 5 feet higher than top-feed furnaces are built in Colorado. There was nothing

unusual about the furnaces except their height and the feeding device, which was in imitation of the bell and hopper used with iron blast furnaces.

On December 26, 1895, one of these furnaces was blown in on a charge containing 14 per cent. lead and a slag of about 33 per cent. SiO_2, 37 per cent. FeO, and 18 per cent. CaO. It started off as furnaces generally do, with all tuyeres bright, slag hot, and producing lead a little too rapidly on account of the displacement of lead in the crucible by slag and matte accumulations.

The furnace was put on full blast about 6 P.M. and ran fairly well until about 4 A.M. the next morning, when the tuyeres had blackened and overfire started. The lead production ceased entirely by 12 o'clock M., and from that time forward until it was shut down the furnace did not produce any more lead. The charge was changed many times; slag and matte charges were fed and the blast was reduced in the hope of getting the overfire down. The bell and hopper feed was perfectly tight, and the furnace was closed up on top as tight as a tin box. The furnace continued to make slag and smelted a fair tonnage of ore, but produced no lead. The writer was of the opinion that the bottom of the crucible had sprung a leak and the lead was going into the foundation.

The furnace was then shut down and the other furnace, which had been put in readiness, was blown in. The charge used did not differ greatly from that put on the first furnace, though the fuel was increased from 12 to 14 per cent. and the blast pressure reduced, but in spite of these precautions and the most careful attention the furnace got hot on top and the lead production stopped entirely in about 24 hours after the furnace was started.

From this point until the furnace was blown out four days later, all the changes that could be suggested were tried. The slag was made acid and then it was made basic. It was run high in lime and low in lime, but no lead came out of the furnaces. Bullion was fed back again and it seemed to disappear as completely as if it had never existed.

The lead gradually fell in the well and bullion was melted in the pot and poured into the well until it was full, when it would gradually disappear again, showing either that there

was no lead going into the crucible or that the crucible was leaking. After a council of war it was decided to blow the furnace out, as too much money was being lost to allow this state of affairs to continue.

Accordingly it was decided that another man should try to run them, and Dr. Charles Harbordt was sent there to do the metallurgical work. The doctor is a valued friend of the writer, and was offered every assistance that the light of past experience could give as to the running of these furnaces. One of them was blown in, and to describe its working would only be a repetition of what has been said of the other two attempts, except that it ran a week instead of four days without producing any lead. The furnace was then blown out and a message sent to headquarters that no more furnaces would be blown in until the general superintendent came down. When he arrived on the scene he blew in one of the furnaces, which produced lead for three days, but in a rapidly decreasing proportion to the amount on the charge.

The second furnace was blown in and acted the same as in all previous campaigns, producing a little lead for two days and then the lead production became nil. The tuyeres would become black and hard, and frequently raw ore could be found at the crucible. Shutting off the blast from the tuyeres would have the effect of burning off the nose after several hours, but it would only take them as many minutes to become black again when the blast was turned on once more. Besides, the furnaces were not being run to keep the tuyeres bright but to produce lead, and as to this point there could be no question—they were a failure.

The bell and hopper feed was withdrawn from the first furnace, and it was again blown in, all the ores and coke being thoroughly soaked in water. Frequently the hose was used with good effect on the feed floor and gradually the zone of fusion was brought down to the tuyeres. The lead production, which up to this point had been nil, gradually rose until about 60 per cent. of the lead on the charge was produced as bullion. This was the best that could be done. When the bell and hopper feed was again replaced the lead production ceased as quickly as if there had been no lead contents in the charge.

To a metallurgist who had not seen these things as recorded

this might seem to be an exaggerated statement. It would appear to be impossible to lose all the lead in the charge if so much as 14 per cent. were used. And that is exactly the way it looked to the writer when the first two furnaces were blown in, but after seeing three more in blast and under the supervision of men who have had long experience in lead smelting, he became convinced that science was indebted to the designer for finding out two things: First, that a lead furnace closed in on top will soon show overfire, and the lead will be wholly or partially lost as volatilized fume; second, that increasing the distance between tuyere level and feed floor beyond the proper height of the smelting column has the same effect as closing the top of the furnace. It assists in volatilizing lead by preventing the cold air which is drawn in above the charge from cooling the top layers of charge.

After making many attempts to correct the losses it finally became necessary to tear out the crucibles, cut off the shaft of the furnace to about the standard height, 22 to 23 feet from the furnace floor to charge floor, throw out the automatic feeding device, and return to feeding with a shovel.

It is certainly of great value to the profession to know what cannot as well as what can be done, and while this information is all of a negative character, still in many respects such an example has its uses. While it was certainly very annoying and perplexing at the time, it is all clear now, and the writer is very glad to be able to impart the experience to others even at the risk of being suspected as a party to the error.

After blowing out, a sample of the bullion remaining in the crucibles was taken, and it was found to assay about 400 ounces Ag per ton, while according to the charge calculation it should only have had 180 ounces Ag, thus showing that the lead had been volatilized in much greater proportion than the silver, and that there had been a cupelling action in the furnace.

After a careful sampling and weighing of all products and giving credit for the lead in the slag produced, which was all high in silver and lead, it was found that 63 per cent. of the lead and 23 per cent. of the silver were unaccounted for.

The deductions that can be drawn from this failure are very simple and exceedingly important. They point clearly

and with force to the fact that the first lead furnaces in use in this country were better adapted to the saving of lead than the modern high-shaft, top-feed furnace. The furnaces with the charge doors on the side, a stack above the floor, with downtake from this stack to the flue beneath the floor, as originally constructed in Leadville and at the Colorado Smelting Co.'s works in Pueblo and also at Great Falls, Mont., nearly fulfill the opposite of all conditions that existed at the furnaces at Aguas Calientes. In the first place, the amount of flue dust and the loss resulting therefrom is much less in a stack furnace than in a top feed, for the very sufficient reason that the flue dust must rise a distance of 10 to 14 feet before entering the downtake of a stack furnace, and, on the other hand, in a top feed the downtake is right at the top of the charge, where every inducement is offered for fines to go into the flue and thus heavily increase the mechanical loss.

In a stack furnace the air entering the charge doors comes into contact with the top of charge, thereby preventing considerable lead losses. That the cooling of the top of the charge in the furnace by the air drawn in at the charge doors has this effect is proved by the excessive losses in the furnaces at Aguas Calientes, where the only point of difference was bad draught and excessive height of shaft above the top of the charge, which prevented the cold air having access to the charge. The downtake was immediately below the charge floor, and the furnaces were fed at various depths from the downtake to thirteen feet below the charge floor. They produced no lead when the smelting column was 18 feet above the tuyeres, and did the best work when it was 10 to 11.

This experience, disastrous though it was, was one of the greatest object lessons that has ever been given to lead smelters. It shows plainly that lead has to be treated as a very volatile metal closely resembling mercury in its behavior, and that if the greatest possible saving is to be made the blast pressure must be light, the smelting column not much over 12 feet above the tuyeres, and the greatest possible amount of cold air must be allowed to enter the furnace at the top of the charge. If these points are well taken, then all the top-feed thimble furnaces that have been built so extensively in the last ten years are steps in the wrong direction.

There is another point about the stack furnace that is much in its favor, and that is, being fully five feet shorter from the charge floor to the furnace floor, it is much easier to bar down when accretions form and, by doing this once a month, can be kept in blast for any length of time desired.

No doubt there will be many to disagree with these conclusions, and the writer will admit that he was of different mind until the force of experience brought out the points in question.

Briefly stated, the points of superiority of the stack furnace over the top feed are:

1st, It makes less flue dust.

2nd, It runs cooler on top.

3rd, It loses less by volatilization.

4th, It is more easily barred down.

5th, It can be kept in blast longer.

At the Arkansas Valley works the furnaces as originally built were of the stack type. They were designed by Mr. Eilers, and the same type of furnaces are now in use at all the works where he has had charge. It has come to be a matter of comment that a man posted in the hobbies of metallurgists can tell by the appearance of a plant who was its designer. Certainly in this case the hobby was of the right sort.

Many kinds of thimbles and charging devices have been introduced and used on lead as well as on copper furnaces, and they have as often been discarded, the method of feeding with the shovel into an open-top furnace thus far being found superior to mechanical feeding for many reasons.

First, the furnace does not run alike in all parts and requires to be fed the greatest amount at the point where it sinks the most rapidly. Second, certain kinds of material must be put in particular places in order to correct irregularities of running. Third, any approach towards closing in the top always has the effect of drawing the fire up, resulting in increased losses by volatilization. Fourth, this loss by volatilization in case of lead will increase rapidly as the air is prevented from entering at the top of charge.

The best method of feeding is to have a fender in front of the charge door (to prevent as much as possible the dumping of charges into the furnace either accidentally or intentionally,

but more often the latter) and to scatter the charge thoroughly over the surface of the coke which has been shoveled in in the same way, but with more on the sides near the furnace walls than in the center. The reason for putting the coke next the furnace walls is that as it goes down it may burn off accretions, and when it has arrived at the tuyeres the heat will be driven into the ore. Sometimes accretions of considerable size can be taken from the walls of the furnace by persistently feeding the fuel against them. From the very nature of things it will be apparent that a chance distribution of the charge, as would be the case with bell and hopper, is the poorest possible contrivance for feeding a lead furnace. It worked on the copper furnaces at Aguas Calientes, though very unsatisfactorily—principally because it was impossible to keep the furnace cool on top, the result being high silver losses and the consumption of the fuel before it arrived at the proper smelting zone.

CHAPTER IX.

Lead Slags and Losses in Lead Smelting.

The calculation of slags for lead furnaces is carried on in exactly the same way as for copper work, with the additional care that has to be taken to keep a sharp lookout for the zinc, sulphur, and baryta. The ores have to be bedded in such a way as to admit of their being used to the best advantage, and this is an absolutely necessary precaution to insure success. Frequently it may be of advantage to change the charge, to put on more or less of some class of ore, either lead, dry or sulphide, and unless the ores are bedded according to these classifications it might be difficult to make the required alteration.

The following is a fair example of lead charge as made up from beds:

	Weight.	Per cent. SiO_2.	Pounds SiO_2.	Per cent. FeO.	Pounds FeO.	Per cent. CaO.	Pounds CaO.	Per cent. Pb.	Pounds Pb.	Per cent. S.	Pounds S.
Carbonate	400	26	104	31	124	6	24	12	48	4	16
Silicious lead	200	41	82	12	24	2	4	10	20	6	12
Roasted	100	15	15	35	35	0	0	31	31	5	5
Iron ore	100	10	10	55	55	2	2	2	2	. .	. .
Lime	200	3	6	. .	. . .	52	104	. .	. .	. .	. .
	1000	. .	217	. .	238	. .	134	. .	101	. .	33

Owing to the presence of some lead and zinc in lead slags, the sum of the SiO_2, FeO, and CaO do not, as a rule, amount to more than 88 per cent., so that figure has been used in this calculation. Also, it has been assumed that 20 pounds of the 33 pounds of sulphur on the charge will go into the matte while the other thirteen will be burned off as sulphurous

acid. Twenty pounds of sulphur would indicate about 80 pounds of matte, as lead mattes carry about 25 per cent. S, and the iron equivalent of 40 FeO, so that 40 per cent. of 80 pounds equals 32 FeO, and that amount should go into the matte, leaving 206 pounds of FeO for the slag.

This type of slag, 34 SiO_2, 33 FeO, 23 CaO, is a favorite with many smelters, and is largely used in the smelting of copper as well as of lead, although in copper work much less lime will do just as well and result in treating more ore. With only 33 pounds of sulphur on the charge the matte formation would in all probability not be as much as 80 pounds, and very likely not more than 50 or 60; if the lower figure should prove to be correct, then only 20 pounds of FeO should be deducted, which would change the resulting slag composition to 33.4 per cent. SiO_2, 34.1 per cent. FeO, and 21 per cent. CaO, which is just as good a slag as the first, and whichever is used, there would be no material difference in cleanness of work. Slags as low as 30 per cent. SiO_2, and as high as 40 per cent. FeO and 20 per cent. CaO, are perfectly safe, and formerly, under certain peculiar conditions which now unfortunately do not exist, were good commercially.

Generally speaking, iron slags are more fusible than lime, and as a consequence give rise to less loss in the volatilization of lead and silver, and they will absorb more zinc without becoming dangerously hard in the taphole.

The peculiar feature of high lime slags is their remarkable flint-like toughness when they once get cold. A furnace running on such a slag may be perfectly free and all right, and owing to a hard breast may have slag running out of half the tuyeres fifteen minutes later.

The writer has in mind one superintendent who if he has one particular hobby more than another it is to run on high lime. No matter what is to be accomplished, it can only be done by raising the lime. If the reduction is to be improved, the fuel cut down, the furnaces run faster, or what not, just raise the lime and, presto, you have it. Another peculiarity with certain persons is that by some optical method of analysis the lime is always higher than a chemist can get by any known method except the "graphite."

It is certainly a bad plan for a metallurgist to insist on

having the slag analyses agree with his calculations, and to insinuate that the chemist does not know what he is about simply because he reports the results as he gets them. It might appear that this is totally superfluous, but the writer has lived in the same atmosphere with a great many men who would forget all the possibilities of mistakes that could be made in weighing charges, in bedding ores, of throwing on the wrong ore at the scales and a shovelful more or less, as the fancy struck them, and then blame the chemist for inaccuracy of work. Besides, slag as it comes from the furnace is not of a constant composition. Every pot is slightly different from every other pot, and a close approximation is all that can be expected.

The losses in lead smelting are generally estimated at 5 per cent. Ag and 10 per cent. Pb, and in making purchases of ore it is customary to make this deduction from the metallic contents of the ore in making payment. The actual losses, or such as are shown in statements, vary according to local conditions of charge, plant, and ability of the metallurgist.

With three unknown quantities in the equation, each of which is subject to great variation, it is not surprising that the known results should vary all the way from a 5 per cent. gain of silver to a 23 per cent. loss, and from a 2 per cent. gain of lead to a 63 per cent. loss. The latter figures for high losses cover the results of the different campaigns at Aguas Calientes, with the bell and hopper feed, and the gains, which were equally remarkable, were obtained at the Arkansas Valley Smelting Co.'s works, Leadville, by the writer while treating a pile of waste material, the contents of which were not charged on the books. It will, of course, be understood that it is impossible to make a gain unless some such conditions as those mentioned are present, where the recovery of something supposed to be lost is counted as a gain.

Properly speaking, the losses in lead smelting can be kept within the allowed limits of 5 per cent. Ag and 10 per cent. Pb, but to do this it is necessary that the furnaces shall not be run on too low a lead charge, say not below 11 per cent., or the tonnage crowded too much by high blast.

The losses of lead in the slag will increase as the amount on the charge decreases, for the reason that there will be more

slag, and with the same contents of lead it would represent a greater percentage of loss. In addition as the lead on the charge decreases the charge becomes more infusible, and as a consequence a greater amount of lead is lost by volatilization.

The writer has had returns from months during which the results on accurate charging of all contents represented a loss of 4 per cent. Ag and 8 per cent. Pb, and in other months 7 per cent. Ag and 14 per cent. Pb. At all of the Colorado smelteries there is an effort to crowd the tonnage of the furnace as much as possible, and this can only result in heavy losses of silver and lead. There is also considerable roasting and slagging of sulphides as a preliminary step to smelting, and it is safe to say that 5 per cent. of the silver and 15 per cent. of the lead contents are often lost in this one operation alone. If the ores are simply roasted the loss will be very much lighter, probably not over 2 per cent. Ag and 4 per cent. Pb, but the slagging or fusing of the roasted charges in the fuse-box, as practiced at many of these works, is certainly productive of very high losses in the general return. Accurate returns from slagging a lot of 50 tons of flue dust at the Arkansas Valley smelting works, in Leadville, showed 11 per cent. silver and 25 per cent. lead loss. Similar tests on slagging ores containing no lead showed 5 to 8 per cent. silver loss in roasting and slagging, while the simple roasting loss was below 1 per cent. If any considerable portion of the tonnage of a smelter is treated in this way, the smelting losses cannot be expected to be within 5 and 10 per cent. limits. There is another feature that plays an important part in smelting losses and that is the amount of by-products or matte, barrings, and flue dust produced. If the contents of these by-products are credited to the smelting account at any more than 95 per cent. of the full amount, an error is made, for they have to be smelted again, and it is safe to say that in doing so 5 per cent. of the silver and 10 per cent. of the lead contents will be lost.

The percentage of fuel necessary to do good work or to smelt different classes of ore at different plants is subject to great variation. The simple fact that a furnace can be run on 9 to 10 per cent. fuel basis is not proof sufficient that the

addition of 2 or 4 per cent. would not smelt enough additional ore to make it a better commercial proposition. For example, at Denver, Pueblo, or Leadville, with coke at $6 and the average charge margin at $4, a furnace that would smelt 40 tons a day on a 10 per cent. fuel charge, would probably smelt 45 tons on 12 per cent. fuel. In the first case it would use 4 tons coke, $24; margin, $160. In the second it would use 5.10 tons coke, $30.60; margin, $180; showing a gain of $13.40 per furnace in favor of higher fuel. This is only true within certain limits, however, because the increase of fuel beyond the limits required for smelting and reduction does not increase the tonnage, but rather decreases it owing to the extra time required for the combustion of the surplus before the charge can come down.

The effect of different kinds of coke on the running of the furnace as well as upon the losses is very marked, although the reason is at times very obscure. There is a coke made at Sabinas, Mexico, which, according to reports of metallurgists who have used it at Monterey and Velardeña, causes an abnormal loss of lead in the blast furnace, owing probably to high percentage of ash and volatile matter; the volatile matter passing off as gas causes overfire, which in turn volatilizes lead. A coke may have as much as 20 per cent. ash and if well baked and free from volatile matter may yet produce fair results in the blast furnace, although a coke with less ash will do proportionately better work.

It is also customary to increase the fuel accordingly as the altitude of the place is higher. At Leadville the fuel consumption is from 3 to 5 per cent. more than at Pueblo, 5000 feet lower. The reason for this is that a greater quantity of air has to be blown through the furnace to get the required amount of oxygen, and the surplus of inert nitrogen produces a chilling effect on the furnace, which has to be remedied by the addition of more fuel.

CHAPTER X.

Improvements in Roasting Furnaces.

In the past five years many improvements in roasting furnaces have been made having for their object the reduction of labor and fuel expense in desulphurizing copper ores and concentrates. The attempt has been made with the Brown-Allen-O'Harra to utilize the heat from the reverberatory smelting furnace to do the roasting, but owing to the breaking of the chain, as well as the interruption of the draft at the doors where the plows pass in and out, it has been found impossible to keep the furnace in constant operation or to develop the amount of heat required. To overcome these defects and to make it possible to roast, smelt, and convert with only one firing and with the minimum amount of labor, the writer has designed a roasting furnace that can be coupled on to a reverberatory and the heat from the smelting utilized for roasting as well as the converting of the matte. The great reduction in the price of steel rails in the last few years is an example of the benefit derived from utilizing heat once developed and not allowed to go to waste before the finished product is turned out. The cast iron is taken direct from the blast furnace to the converter, and the converter blooms run hot into the soaking pit to await the rolls, where they are rapidly worked up into steel rails without having lost the heat imparted by the coke of the smelting furnace. If a great economy in the cost of producing copper is to be effected much the same policy must be pursued, and this design is here offered to show what can be done in this direction. (Figs. 16 and 17).

Of all the mechanical roasting furnaces yet designed the O'Harra is the most suitable to be connected with a rever-

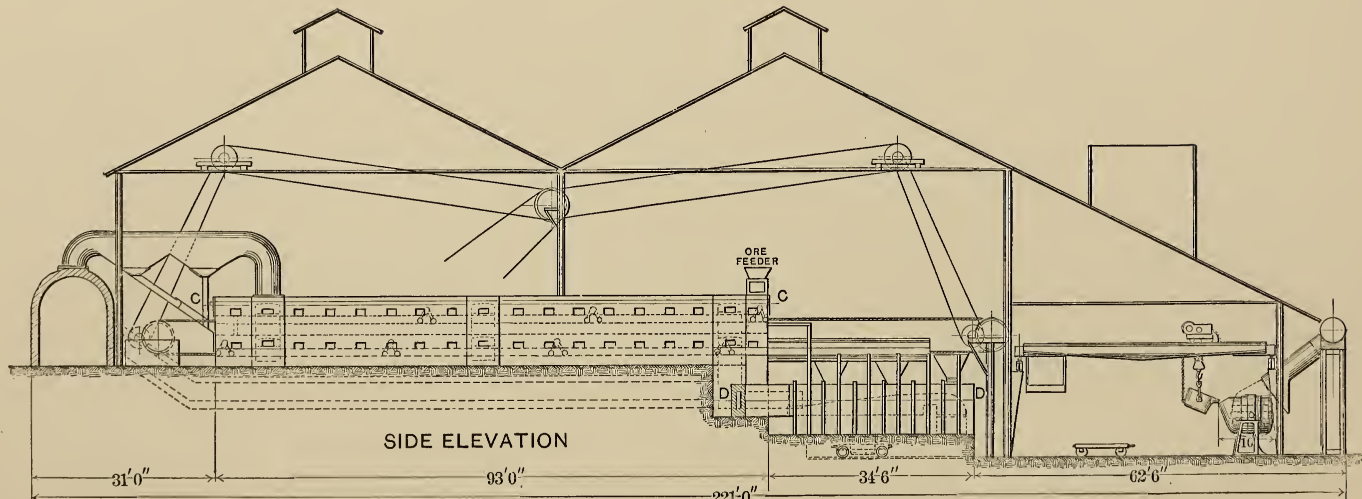


FIG. 16 – PROPOSED COMBINATION OF ROASTING SMELTING, AND CONVERTING PLANTS; ELEVATION

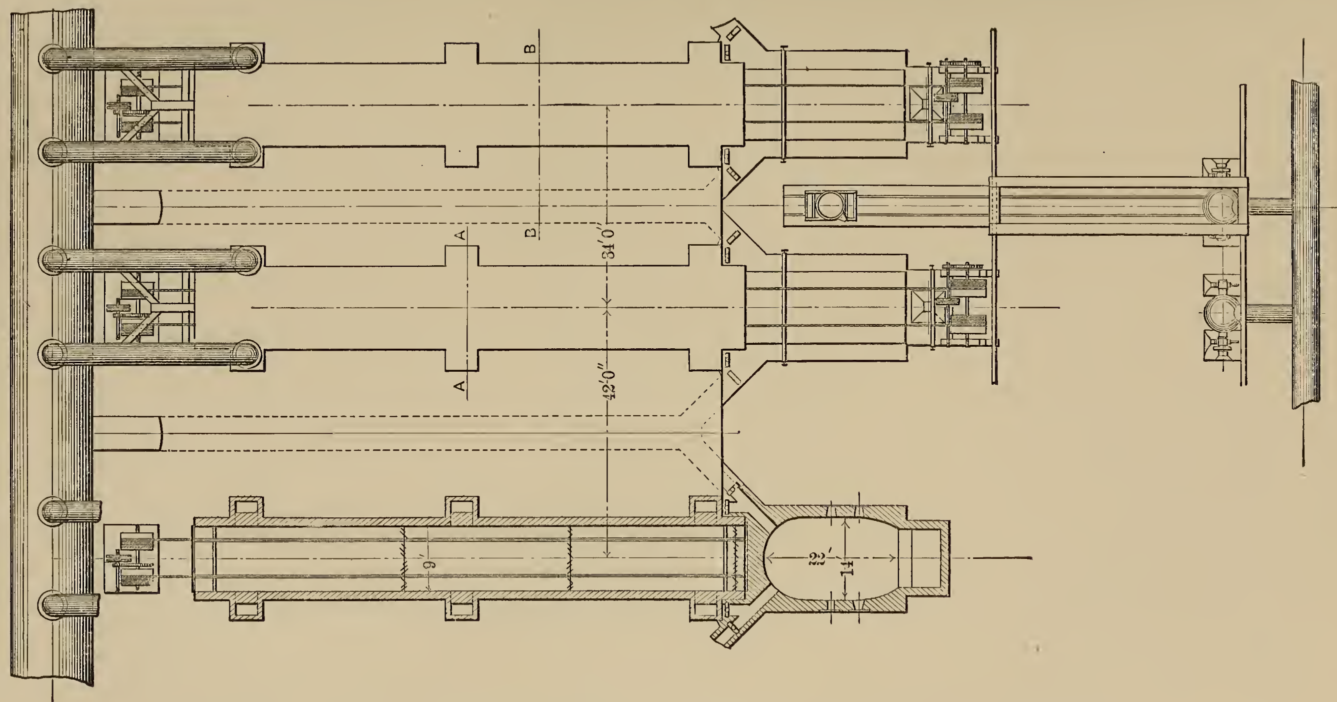


FIG. 17.—PROPOSED COMBINATION OF ROASTING, SMELTING, AND CONVERTING PLANTS; PLAN.

beratory furnace on account of its long hearth, and the fact that it can be made to deliver the calcines into hoppers on top of the smelting furnace. The difficulties have been with the chain, which was subject to much wear on account of passing through the fire, and the opening of the flap-doors on the ends of the furnace each time a set of plows passed in or out of the furnace. The flap-doors let in so much air that the hearth was cooled for several feet near each end and the draught of the furnace seriously affected.

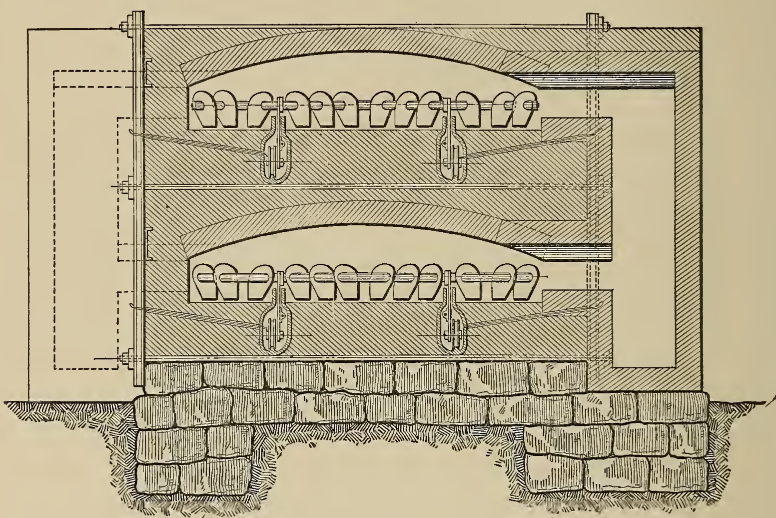


FIG. 18.—THE HIXON ROASTING FURNACE; CROSS-SECTION.

To obviate these drawbacks, and especially to remove the chain from the fire, the furnace illustrated in Figs. 18 and 19 was designed. Two cast-iron conduits are built into the hearth of each floor, and the wire rope as well as the wheels which carry the plow arms move in these conduits, nothing but the plows and the arms being above the slot, the action of the plows being similar to that of a planer in a machine shop. They move forward and backward on the same deck without passing out of the furnace, and plow in only one direction. A knot on the rope striking the reversing gear throws out the clutch at one end and at the same time throws in a clutch at the other end of the furnace, so that the motion

is reversed automatically each time the plows reach the end of their path. By means of the lever arm shown in the drawing attached to the rope and turning on the plow arm, the plows are alternately thrown in and out of action according to the direction of their travel. The plows traveling a greater distance than they are placed apart, the ore is picked up and carried along by one set after another has dropped it until the end of the stroke, when the next set of plows comes back of the place before taking hold and carries it along to the opening through which it falls to the lower hearth. Here the

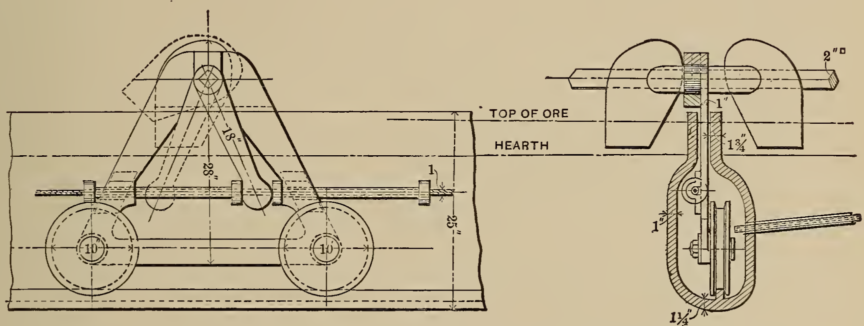


FIG. 19 —THE HIXON ROASTING FURNACE; DETAILS OF CARRIAGE AND PLOWS.

plows act in the opposite direction and the calcines are finally delivered into the hoppers over the matting furnace ready to smelt. The lower deck is extended on I-beams out over the matting furnace, and is covered in so that the ores will continue to be as hot as when in the furnace. By the action of the draught cold air is drawn in through the conduits and fed up into the furnace, at the same time cooling the rope and wheels.

The plows may be removed from the trucks by drawing them out through the removable door at the end of the furnace and detaching the rope from the lever arm. Another set can be substituted and work go forward with but short delay. As the plow arm is supported at two points and is drawn by two ropes, it is capable of being made much longer than the O'Harra plow arm, and the furnace hearth can be built any width or length desired, with either single or multiple decks.

The heat from the smelting furnace is taken out of the two

corner flues to either side of the roasting furnace and led into the different roasting hearths on the side, where it passes along to the other end and unites again to pass out through the iron flues, in which would be deposited the heaviest of the dust. The dust from these hoppers can be conveyed by a spout to the lower hearth and taken by the plows to the smelting furnace. The lateral flues and dampers shown in the drawing are provided so that any portion or all of the heat from the smelting furnace may be sent through or around the roasting furnace to the main dust chamber. This si necessary for the proper regulation of the temperature in the roasting furnace as well as for emergencies when the plows need changing. The slag from the reverberatory can be skimmed into a ladle and taken away by car or granulated by water, while the matte would be poured from the ladle into a converter and blown to copper.

CHAPTER XI.

Smelting Raw Concentrates with Hot Blast at Anaconda.

The experiment of smelting raw concentrates with hot blast was tried, and while the results were not satisfactory they were nevertheless instructive. Two brick stoves, similar in construction to iron furnace stoves, were erected by using sections of old Bruckner furnaces, which were bolted together to form the shell. They were built 45 feet high, 8 feet in diameter, and were lined with a checkerwork of firebrick made in Anaconda.

The fuel used was slack coal in Taylor gas producers. The gas was burned in one stove while the blast was going through the other. The blast could be heated to about 500° F. A furnace was run for some time on a charge of 1000 pounds raw concentrates and 1000 pounds converter slag. The matte produced contained about 35 per cent. Cu (too low for converting at Anaconda), the concentration effected being about three into one.

In the resmelting of the matte with hot blast, together with converter slag, the grade was only increased to 45 per cent. Cu, showing that after the copper has reached a certain point hot blast fails to cause the elimination of iron. About 5 per cent. fuel was used during the experiments, but later this was increased to eight and hot blast was used in smelting the refuse from converting. It effected a saving of fuel as against another furnace run on cold blast, until by the alternate expanding and contracting the brick lining of the stoves gave out and the experiment was abandoned. The saving by the use of hot blast was about $25 per day, but the stove linings were too expensive to admit of any economy even at that rate.

CHAPTER XII.

Copper Converting at Anaconda.

At the time the writer entered the employ of the company for the purpose of constructing a converter plant to treat the entire tonnage of matte produced, about 300 tons daily, there was in operation a converter plant of twelve vessels that had been erected as an experiment. The workings of this plant had been very unsatisfactory, and the short life of the linings as well as the large losses had caused it to be severely criticised. At that time the process of copper converting was not very well understood, and it was the vain hope of many to find a lining that would last a week, as in the case of steel-making. That has proved the great stumbling-block for many, though the difference in the processes is apparent. In steel-making the charge is blown only long enough to burn out the excess of carbon, and before the iron has begun to oxidize the converter is turned down and the charge poured out. All of the products of combustion in this case are gaseous except what SiO_2 may be formed by the silicon present in the cast iron. This SiO_2 has a protecting influence on the lining of the vessel, since it will combine with any oxide of iron formed and supply SiO_2 that would otherwise have to come out of the lining. In copper converting the matte may contain from 13 to 35 per cent. of Fe in the form of sulphide, all of which has to be converted into the oxide before it can be separated from the copper and sulphur. The 13 per cent. of iron would indicate a matte of 60 per cent. Cu, and the 35 per cent. of iron about 28 per cent. Cu, these being the limits within which the writer has converted.

In the case of a 60 per cent. matte with 13 per cent. Fe, each ton of matte charged into the converter would contain

260 pounds of Fe. A partial analysis of the slag formed after the charge has been blown to the skimming point is SiO_2 37, Fe 38, Cu 5, showing that for each pound of iron in the matte a pound of silica has to be provided in order to form a slag which will, with the limited heat generated by the combustion of the Fe to FeO and the partial combustion of the sulphur to SO_2, remain fluid and admit of being poured out of the vessel. It must be borne in mind that it is not only a question of oxidizing the iron, but of separating it from the charge after it has been oxidized and removing it from the converter as quickly as possible. This can only be done by forming a fusible slag, and a slag can only be formed by the union of an acid and bases, and as the bases are formed naturally in the converter the acid must be present to unite with them as fast as they are formed, otherwise there will be an accumulation of very infusible FeO in the converter which will mix up mechanically with the white metal (sulphide of copper) and make a spongy, viscid mass which cannot be skimmed or finished to copper. Bases do not unite to form slags at any practicable temperatures, consequently a basic lining is out of the question for copper converting. The oxide of iron alone without silica will not form a fluid slag, so that if it were possible to run a charge in a water-jacketed converter without freezing, which it is not, it would still be impossible to remove the FeO from the converter because it would not separate from the sulphide of copper.

All chemical unions are attended with the development of more or less heat, and it is quite probable that the union of FeO with SiO_2, to form a slag, is attended with a considerable heat development, and if the converter were robbed of this by the use of any other kind of a lining the heat would be insufficient. As for basic lining and water-jacketed converters, the writer has tried them both and will give in detail the results so that aside from theory the actual outcome may be known.

The first experiment to be tried was to protect the lining immediately above the tuyeres where the corrosion is greatest. This was attempted by imbedding in the silica and clay lining a pipe coil running around three sides of the converter and immediately above the tuyeres. This coil was of 1¼-inch

pipe, put together with malleable ells and clamped to the shell of the vessel by stay bolts about 8 inches in length. The coil was connected at the inlet by a hose to the water main in which the pressure was 45 pounds; the discharge was also provided with a hose connection so that the converter could be turned up or down without interfering with either. The lining was put in in the usual manner and thoroughly rammed behind the pipe coil, and as the lining was about 20 inches thick at the tuyeres the charge could not come into contact with the coil until the 12 inches of clay covering had been eaten away. The first and second charges were run as usual with the rapid decrease in the thickness of the lining and increase in size of the vessel. The third charge cut away all the lining in front of the coil, and when the converter was turned down to skim slag the pipes could be seen bare and exposed from the nose of the converter. At this point the experiment was attended with considerable danger, for if the pipe had given way or been attacked by the matte an explosion would certainly have occurred that would have wrecked the entire plant as well as killed the men working on the vessel. For this reason everybody kept at a respectful distance while the converter was blowing, and only approached when necessary to turn the vessel up or down. But nothing having occurred to frighten the men they gradually came to regard it as being safe and returned to work as usual.

The charge was finished and the copper poured when it was found that large lumps of copper were adhering to the tuyeres and pipe, but at other points the lining had been corroded as much as usual. A fourth charge was attempted, but the bottom of the vessel became too thin and the charge was lost on account of breaking through at that point.

It was thus demonstrated that protecting the lining at one point only changed the corrosion to another, so that it became a question of abandoning the experiment or putting in pipe enough to water-jacket the whole interior. This was done, and a coil was constructed and put into the converter which should be imbedded in and protect the lining as high as the top of the charge. The converter was charged and the first charge finished without exposing the pipe; the second charge

was blown to slag and skimmed but ran cold on the finish blow, and more matte was tapped in and blown to slag and skimmed again. By this time the pipes were exposed throughout the interior of the vessel and the lining had been eaten away from behind them in some places. The attempt to finish the charge was a failure, and it had to be poured out as white metal and used as scrap in the other converters. Many other attempts were made with this converter, jacketed as it was, but they all ended in failure and demonstrated beyond a doubt that a charge could not be blown to slag after the pipe became exposed, for the reason that there was no silica to flux the basic FeO formed by the oxidation of the iron in the matte. Water-jackets were then abandoned and a lining composed of burned lime mixed with coal tar was tried. The result was a foregone conclusion, but nevertheless we were hunting for straws to grab at and it was given a fair and impartial trial. Just enough tar was used to stick the lime together, and when in place the lining was baked with a coke fire to drive off the gaseous portion of the tar. A charge was run into the vessel and blown until the flame at the nose indicated the complete oxidation of the iron. The converter was then turned down to skim, but in place of fluid slag and matte there was found an indescribable mush of matte, lining, and oxide of iron, with which nothing could be done, and after one more trial with the same result the experiment was abandoned.

In conclusion, and before leaving the subject, it is well to state that the silica lining for a copper converter serves a double purpose, that of a lining and also a flux, and that it is necessary to the success of the process that the lining should be corroded. If this corrosion were stopped by any means except the introduction of silica in some other way, the process would be defeated. The attempt to introduce silica through the tuyeres with the blast has thus far proven a failure, and very likely always will, for mechanical reasons. In the first place the time during which SiO_2 is needed is very short, and the quantity required is too large to admit of even a small part of it being introduced.

Take the case of a five-ton charge of matte containing Cu 35 per cent., Fe 30 per cent., S 26 per cent. Each ton of

matte would contain 600 pounds of Fe, and the five tons would require 1,620 pounds of SiO_2 to be introduced through the tuyeres in the space of one hour in order to make a slag with 25 per cent. SiO_2, and 60 per cent. FeO.

The lower grade in copper the matte is, and the higher in iron, the more basic will be the resulting slag, and while 55 to 60 per cent. Cu matte will make slags as high as 40 per cent. SiO_2, a 35 per cent. Cu matte will make a slag with about 25 per cent. SiO_2, and 60 per cent. FeO.

The rapid corrosion has the effect of making the lining lose its binding force and it falls off in chunks, just as the rapid erosion of the banks of a stream by water will make them cave off in large pieces. Consequently the life of a lining is not exactly inversely proportionate to the amount of iron in the matte, but decreases in length more rapidly than the iron increases. For example, a lining that would produce 200 bars on 55 per cent. Cu matte with an iron contents of 17 per cent. would probably not produce more than 50 on a 28 per cent. matte with the Fe 35 per cent. But there are so many chances for weak points to develop unexpectedly in the lining with the low-grade mattes that no comparison is of much value.

After testing all the different schemes that could afford a possible solution of the difficulties of relining converters in the stands, it became apparent that the proposition was a mechanical one; to handle the converters as quickly as possible and make a change of vessels in the least possible time when a lining was destroyed. The practice at the old plant at Anaconda, and for some time at the Parrot, was to run the converter until the lining became too thin to stand another charge, and then to flood the vessel with water from a three-inch hose and allow the stream to run until the temperature had been reduced so that after the water had been turned out a man could go inside, cut out all the loose jagged points and slag shell, and then reline the vessel. The cutting out was done while the vessel was turned bottom up, and the loose material was thoroughly cleaned out before any more lining was put in. The converter was then righted and the lining passed in with a shovel through the nose in lumps, about 8-inch cube. The liner first put in the bottom by throwing the lumps of lining against the bottom of the converter and after-

wards pressing them in place with his foot. The bottom should be from 4 to 6 inches below the tuyeres and made as thick as the side lining. The bottom finished, a circular board about the size of a barrel-head was put on top of the clay for the liner to stand on, and with this as a form the side lining was built up against the old lining remaining in place, or against the shell of the converter if it had all fallen out in the cutting-out process.

Large mitts were used by the liners, and a trowel to cut off and shape the lining after the lumps were pounded into position with the fist. Attempts to use all kinds of tamping irons were made, but it was found that it took much longer to put in a lining with them than with the hands, and there was no apparent improvement in its character.

The lining was made of pure white quartz crushed to the size of slack coal in crushers and rolls and afterwards ground in a Chilian mill with one shovel of fat, sticky clay to 8, 9, or 10 shovels of quartz, according as the clay seemed to vary in plastic or binding qualities. The clay which was employed contained a somewhat large percentage of alkali earths as well as iron, and only so much of it was used as was necessary to stick the quartz together when moistened and ground

CLAY ANALYSIS.

SiO_2	66.0 per cent.
Al_2O_3	18.5
Fe	3.1
CaO	2.9
H_2O	8.4
	98.9

in the Chilian mill. On account of the flooding of the converter before relining, very little difficulty was experienced in making the new lining adhere to the old, but later, when the new plant was constructed and the converters were relined without filling them with water, it would sometimes happen that the fresh lining would part from the old, and linings put into a dry vessel did not last as long as those put into one that had been made thoroughly wet. Notwithstanding this small advantage, the use of water for cooling the vessels is a poor policy for many reasons, the first of which is that there are formed by the blowing of the charge sulphates of iron and copper, which are dissolved by the water and react on the

ironwork of the converter, corroding it rapidly and causing it to break open at the riveted seams after a couple of months continuous run. The lining remaining in the vessel swells by the addition of water and brings a heavy strain on the converter, frequently causing splits of the seams. Moreover all the copper that is dissolved as sulphate is carried away and is lost unless recovered on scrap iron.

At this plant, which, as stated, was an experimental one, there were three converters to each cupola or melting furnace, and the charge was run from the cupola to the converters in long spouts. One of the three converters was kept in operation while one was drying out and the third was relining. By putting on extra crews when the matte was of good grade it was sometimes possible to work five or six of the twelve vessels at one time. But this was only possible when the copper in the matte was as much as 60 per cent. and the iron as low as 13 per cent. Otherwise the linings would be destroyed too rapidly to admit of runing more than one vessel out of the three. Owing to the bad dust-chamber arrangements and to the use of water in the converters, the losses at this plant were exceedingly high. During the year and a half that the writer ran it, before the new plant could be constructed, the losses in copper were 4 per cent. and silver 5 per cent., while the cost of converting varied between 0.72 and 1.34 cents per pound of copper converted from matte averaging 55 per cent. copper.

The converters were 60 by 60 inches, square in section, and turned by a worm gear operated by power from a line shaft. The plant was put into a building which it did not fit, or rather the building did not fit the plant, and, generally speaking, everything worked at a disadvantage. The experience gained was the basis for constructing the new plant and was valuable from that point of view. The largest production in any single month from this plant was 5,500,000 pounds of copper.

Previous to constructing the new plant several systems of handling converters were developed and their merits discussed. It was finally decided that the handling by crane was preferable to the car, and the detail plans were drawn up accordingly.

It was unfortunate, though unavoidable, that the plant had to be built with remelting furnaces, the smelting works being already established and so compact in arrangement that it would have been impossible to take the matte from the reverberatories to the converters in a molten condition. This is a point that in any new construction should be kept in mind, even though converters are not to be installed at first. The extra expense of remelting matte will be about $2 per ton where coke is $12. There is a small loss by flue dust in addition, so that on 50 per cent. matte the remelting expense would be $4 per ton of copper produced, or 0.2 cent per pound of copper converted, which is approximately the difference of cost in favor of a plant where the matte is taken directly from the smelting furnaces to the converters.

This cost would be much less in a locality where labor and fuel are cheaper than in Montana, but in most places in the West the figures would apply, and the cost per pound of copper would also increase rapidly as the grade of the matte fell below 50 per cent., or the tonnage decreased. For instance, when the tonnage handled at the converter plant was small the items of general expense were just as much as when it was large, and would greatly affect the result for that month.

The average cost for converting 55 per cent. matte may be stated at 0.65 cent per pound of copper, divided as follows:

Remelting matte	.2
Labor and lining for converters	.25
Labor on converters	.1
Resmelting converter slag	.05
Supplies	.05
	.65
For a plant without remelting	.45

The losses in converting as shown at Anaconda were about 3 per cent. of the copper contents, but there was a wide difference between assays of samples of matte taken at the converter plant and at the smelter. This difference amounted to 0.5 per cent. on all the matte received, and represented about 1 per cent. of the loss, so that the real loss was probably only 2 per cent. of the copper.

The silver loss was apparently high according to assays of the converter copper, but the casting showed a corresponding gain, and after making allowance for this the loss of silver in converting was reckoned at less than 1 per cent.

CHAPTER XIII.

Blowing a Converter Charge.

The operation of blowing a charge in a converter is one requiring much experience and attention on the part of the skimmer, and can only be learned by actual practice in the business. A knowledge of the changes of flame coloration at the nose, indicating the condition of the charge, is only attained after the apprentice has given much attention to it, and in some cases where color-blindness may exist it is impossible for him to do so at all. The first portion of the blow is usually from 40 to 60 minutes' duration, although if the charge be too heavy it may be prolonged for much longer time. The flame is a light green color, with an occasional shade of yellow in the first stages, probably due to volatilized sulphur. As the time for skimming approaches occasional flashes of azure blue may be seen mingling with the light green, and if sufficiently prolonged it will become wholly azure blue. The converter must be turned down and the blast shut off before this change in the flame coloration has gone too far, or the entire contents of the converter will frequently be blown out and scattered over the building. The writer has seen charges weighing several tons foam and shoot matte thirty feet in the air from being overblown only a few minutes.

The cause of these explosions is the oxidation of a portion of the copper which enters the slag, and when the vessel is turned or the equilibrium of the slag and matte disturbed, it results in the mixing of slag containing oxide with the white metal below; the sulphur of the white metal has a reducing effect on the copper oxide and the result is a sudden precipitation of copper and the formation of large volumes of SO_2 gas, which causes the charge to foam and at times throw the

greater portion of it out of the converter. This may happen before the converter is turned down owing to the agitation of the charge by the blast, but it is most active while the vessel is moving. It may happen after the charge has been skimmed, but in this case it is usually not so serious and shows that the skimming has been done too soon or before the iron has all been oxidized, that portion of the iron remaining in the matte having formed slag, which has absorbed copper oxide from the charge below and has been acted upon by the sulphur in the white metal. It is frequently necessary to skim a charge twice in order to remove the slag formed and take some of the burden off the blast. This will allow the blast to penetrate the charge faster and save time, as well as avoid the possibility of any overblow. It also equalizes the charge burden in cases where several converters are furnished blast from the same pipe.

The formation of copper oxide will not take place to any great extent as long as iron is present in the matte, but as soon as the iron is all oxidized and gone into the slag as silicate, then copper begins to oxidize rapidly, and if the slag is not taken off before this begins it will unavoidably result in a large portion of the slag being thrown out. After the slag has been skimmed off the copper will be oxidized just the same, but so long as there is sulphur present in the charge the reduction goes on just as rapidly, so that there is a constant precipitation of copper and liberation of SO_2. It occasionally happens with the best of skimmers that they will overblow a charge slightly and have slag shooting all over the building, but there is a sharp rivalry between them, and a rude sort of justice is meted out to the offender by jeering and what is commonly called the "horse laugh."

The writer has personally blown and skimmed many charges and has had just such experiences, and has learnt that many of the men who are able to handle a charge perfectly do not know the reason why certain things occur, but they are sure nevertheless of what will occur if nature's rules are violated.

The slag should be poured off as quickly as possible and in a steady stream without moving the converter any more than is necessary, otherwise considerable matte may escape. The slag should flow easily and in a solid, dense stream, but if

matte is escaping with it the bottom or back part of the stream will be seen to vibrate while the slag itself will be less fluid, flowing with the viscosity of molasses. At this point the rabble is shoved into the stream, and if the small crust of slag cut away shows matte the pouring is stopped and the remainder of the slag is skimmed off. Some slag will still remain behind, but the skimming should in all cases be as clean as is practicable under the circumstances. It sometimes happens, especially in large converters running on high-grade matte, that the slag will be granulated or only partially liquid. This may be due to a cold converter in case it is the first charge, or to the lack of heat developed by the charge due to slow running, which latter may be due to low blast or too large a charge. In such a case the remedy depends upon the cause. If it is due to a cold converter the addition of from 2000 to 5000 pounds of matte will in most cases cause the slag to become liquid. If the charge is too heavy the blast pressure must be increased or a portion of the slag skimmed off in the best way possible, and the finish blow made without skimming clean. The unfused material can be smelted by a small tap of matte after the copper is poured off, the slag poured and another tap taken to furnish metal enough to finish. With very low-grade matte several taps have to be taken to get the required amount of white metal to finish. At Aguas Calientes the converter was eight feet in diameter, the largest copper converter in the world, and on low-grade matte a charge of 8000 pounds for the first tap would be taken and blown to white metal, the slag poured off, and a second tap of 10,000 pounds poured in and blown to slag and skimmed in the same way. A third, and sometimes a fourth, tap would be skimmed before the resulting white metal would be sufficient to stand above the tuyeres until the charge was finished. In this way charges of 45,000 pounds of matte were not uncommon, and the resulting copper would not be more than 40 bars of 200 pounds apiece. With matte as low as 30 per cent. the converter increased in size so rapidly, owing to the corrosion of the lining, that it was difficult to get a charge to finish. After the vessel had finished the last charge that the lining would stand, a tap of from 10,000 to 15,000 pounds of matte would be put in and blown to slag, or until it was in danger of coming

through the side or bottom. This was for the purpose of washing out the copper adhering to the lining after the last charge. The resulting white metal was dumped and used as scrap on subsequent charges.

A considerable quantity of cold matte of high grade is necessary after the skim to keep the temperature of the charge from rising too high. If the charge is too hot the oxide of copper formed is not reduced before it forms a silicate with the lining, and as this is very infusible it is chilled by the blast on the tuyeres, resulting in clogging them and making the charge run slow and necessitating much punching. It seems paradoxical, but it is nevertheless true, that above the correct temperature the charge runs much slower and the tuyeres are hard to keep open. By the addition of white metal in large lumps the difficulty can be avoided.

The use of low-grade matte to cool the charge results in the formation of more slag owing to the presence of iron, and this slag will, if the quantity is large, result in slow running and shooting out of the vessel as previously mentioned. If no scrap or white metal is to be had it is sometimes necessary to throw in a bar of copper to bring the temperature down. The question of temperature must of necessity be told by the appearance of the flame at the nose, and is quite as important as the other indications of slag and finished copper.

In the first part of the blow, or before the slag has been skimmed off, it is not so much a question of temperature as how the converter is taking the blast. If the flame from the nose is the full size of the opening and appears to be leaving the converter with considerable velocity, it indicates that the tuyeres are open and the charge is working rapidly and satisfactorily; but if it goes in puffs and has a choked and irregular movement, the tuyeres need to be punched until they are free from obstruction. The tuyeres should always be punched before a charge is put in so that the blast may enter as freely as possible from the beginning. If the charge is sufficiently hot when it enters the converter it will need very little punching, and will start off with a very dense discharge of smoke and considerable volatilized sulphur. Shortly afterward, probably fifteen minutes, it will have slowed down and will need punching for the next five or ten minutes, when

owing to the increased temperature it will run freely until time to skim. If the charge is cold when it enters the converter, punching will be necessary from the beginning and for a longer time thereafter than if it were hot. After skimming the scrap, white metal should be thrown in, and any cleanings that may be on hand should be introduced to the extent of about 10 per cent. of the weight of the charge if the slag is liquid and hot, but in less quantity if it is viscid, and the blast should be turned on as quickly as possible. Punching will be necessary if the flame indicates that the converter is not taking air properly. As long as the flame continues to be voluminous and leaves the nose freely all may be well, but if it assumes a bright brassy color and slows down the charge is probably becoming too hot, when more scrap should be added according to the requirements, and the tuyeres punched until the proper action is restored. If the flame assumes a light orange color that gradually turns into a darker shade, and then takes on a copper bronze color, the indications are that it is rapidly approaching a finish, when very close attention is necessary to avoid an overblow and an oxidized charge.

Just the proper point to turn the vessel down is reached when the little particles of copper ejected from the converter give the appearance of very fine gauze or lines of a copper color, and when coming in contact with any obstacle they cease to adhere, as they will continue to do so long as they are of matte. The copper adhering to the punch-rod will also indicate very closely the time of finish.

The vessel is then turned down and the granulated slag which will be present on top of the charge is shoved aside by means of the rabble until the surface of the liquid copper is exposed. If on skimming off a clean surface the charge shows a bright metallic mirror of copper, the charge is poured into moulds, but if the surface is covered by a skin of black sulphide of copper it is necessary to turn on the blast again for a short time.

If considerable slag is present, or if the charge is slightly overblown, the copper will be covered with a layer of slag which will foam and bubble and require some time and considerable cold slag or cleanings from the floor to chill it

around the nose before the copper can be brought to view. If there is sufficient copper in the converter to stand above the tuyeres it is possible to completely oxidize a charge, but although this sometimes does happen, it is through carelessness or a mistake in judgment on the part of the skimmer. I have known it to happen when copper oxide on a charge already slightly overblown was mistaken for matte and the charge was kept working long after it should have been poured. Such things will happen, but seldom more than once to the same man, and it is not by any means a sign of incompetency, and having once occurred will improve the future service of the skimmer.

If towards the latter portion of the finish blow the flame becomes very dark and red the charge is becoming cold, and the probabilities are that it will not finish without the addition of more matte. If it does finish cold it will be difficult to pour and will leave much copper adhering to the lining. In such cases another small tap of matte is put in and blown to slag and skimmed, or at times when the matte is unusually low it is necessary to throw in a large quantity of cold scrap for the purpose of chilling the slag and making it impossible to skim. The reason for this is that with low-grade mattes the corrosion is so rapid and the addition to the copper contents of the charge so small, that instead of increasing the height of the charge above the tuyeres it is probable that it would be decreased if the slag were poured off. So that it becomes necessary to granulate the slag and force it to mix with the copper, raising the charge above the tuyeres in order to insure the blast penetrating it until the copper is finished.

The methods that are used to develop heat in a charge that has run cold are, first, the addition of billets of wood; heavy cordwood is preferable on account of its greater density and the fact that it will, by floating in the charge and by the gas generated, raise the charge level so that the blast will continue to penetrate the metal bath and develop heat instead of blowing over the top and freezing it. Second, the addition of lump coal, the effect of which is the same as cordwood. Third, the addition of a small amount of matte, finishing with granulated slag as described above. The reason for granulating the slag is that in this condition it will not cause

the shooting and foaming of the charge described when overblown. This last method is, as before stated, employed in case the matte is of low grade, but if it is high grade, say 55 per cent., the slag may be safely skimmed off, the resulting white metal being enough to compensate for the extra corrosion of the lining and also to make up the shortage previously existing. If the converter is of the cylinder or Leghorn type, it can be turned back until the tuyeres are brought to a lower level and the blast forced to penetrate the charge, but with the great majority of converters this is not possible, since turning the converter back beyond a fixed point is prevented by the smokestack into which the fumes are discharged. Only the experience and judgment of the skimmer is to be relied on in such cases, and the remedy must be applied which is best suited to the conditions.

The relative heat-developing power of sulphur and iron is very strikingly illustrated by the action of different grades of matte. A charge of low-grade matte will smelt fully half its weight of granulated slag left in the converter, while a charge of high-grade matte will only add to the difficulty.

The low-grade matte containing more iron has greater heat-developing power, as well as more basic action on the silicious slag, and will bring it to a perfectly fluid condition, while the high-grade matte containing less iron and making a more silicious slag will be unable to smelt the accumulation in the converter. The heat developed in the first portion of the blow, as well as the fact that the iron is all converted to oxide much sooner than the sulphur, shows that iron has a stronger affinity for oxygen than sulphur, and in combining with oxygen developes more heat than the latter. This is also proved by the action of the charge in becoming rapidly hotter in the blow from matte to slag and gradually colder after the iron has been oxidized and the slag skimmed off. It is a common error often repeated that the heat developed in copper converting is due entirely to the burning of the sulphur, while the fact is that the more heat is developed by the burning of the iron.

For this very reason it becomes difficult to convert mattes as high as 65 per cent. Cu on account of the decreased amount of iron and insufficient heat development. It is apparent at

once that the presence of the iron is necessary, and that, being present, silica must be provided in the lining for it to act upon to form a fluid slag, and, further, that if the corrosion of the lining is stopped the process will be defeated. The silicious lining is as much a part of the process of copper converting as magnesia lining is in the basic Bessemer treatment of phosphoric iron, and it is suicidal to attempt any other kind of lining, either water-jacketed or basic. The improvement, if there is to be any, is to be in the line of mechanical devices, and the use of silicious ores to replace the expensive quartz and clay linings.

At Aguas Calientes the lining was made entirely of ore, and this contributed a great deal to the success of converting low-grade matte at that point. If quartz had been used instead of ore the expense would have been too great to admit of 30 per cent. Cu matte being converted. It was very fortunate that an ore of such ideal composition for the purpose was to be had. This ore came from Pachuca, State of Hidalgo, and was mined there in large quantities.

A partial analysis showed: SiO_2 72 per cent., FeO 5 per cent., CaO 0.6 per cent., Al_2O_3 15 per cent. The ore ground in a Chilian mill with water was very plastic and did not need the addition of clay, and it was possible to run with a lining on which there was a margin of $20 (Mexican) per ton. If such ores could be obtained in Montana it would prove a bonanza to the copper converters.

There are three kinds of finish on converter copper, according to the time the charge is turned down.

The first shows a small amount of regal and is usually about 95 per cent. Cu, and sometimes expands to such an extent on cooling as to make it exceedingly difficult to get the bars out of the moulds. On this account, as well as the extra time and expense in refining before casting into anodes, it is seldom made, and then only when a charge is too cold to be kept longer in the converter, or on account of weak lining.

The second and most common and desirable of the three is called gas finish, on account of the large quantity of SO_2 which leaves the metal on cooling. This finish shows no regal but contains SO_2, dissolved in the copper to such an extent that a mould filled with the metal will, after the ebullition has

ceased, not be more than one-third to one-half full, and it is necessary to pour into the moulds two or three times in order to make a fair-sized bar.

The SO_2 will remain with the copper as long as it is in a molten condition in the converter, but as soon as it strikes the cold mould and begins to solidify, the gas comes off rapidly, and if great care is not taken in pouring into the moulds, the copper will effervesce and run over like soda water.

At times a crust will form over the top of the bar before the gas has escaped from the liquid interior, and then a rupture will take place and a stream of molten copper may be thrown a distance of several feet by the escaping gas. Serious burns frequently and unfortunately occur on this account, and it is rather dangerous to stand near the moulds until the copper is thoroughly solidified. There seems to be a very strong resemblance between the affinity of molten silver for oxygen and this peculiar action of converted copper and SO_2. It does not occur with blister made in reverberatories for the reason that all the oxidization takes place on the surface, while in the converter it goes on much faster and all through the metal bath, some of the SO_2 formed being dissolved in the copper.

The third is called blister finish and exhibits the characteristic blisters on the surface of the bars from which it gets its name. In a charge of ordinary size there is about ten minutes' difference in time between the first and second finish, and about five minutes between gas finish and blister finish. The blister finish contains a small amount of gas, but usually not enough to cause the copper to decrease much in volume on cooling. It is seldom that a whole charge, especially if it be a large one, will be blister finish. Usually the last few bars are gas finish, while the copper first poured from the top of the charge will be blister. In order to produce blister it is necessary to overblow slightly, and some copper oxide will be floating on the surface as slag.

The distribution of silver in the copper bars, as determined by assaying samples taken from different parts, shows the folly of trying to get a correct sample except after remelting or from the stream as it comes from the converters. The results of two bars sampled at Aguas Calientes are given below:

Blister Finish.			Gas Finish.		
No.	Part of bar.	Assay in troy ounces silver per ton.	No.	Part of bar.	Assay in troy ounces silver per ton.
1	End,	358.1	1	End,	231.2
2	End,	403.4	2	End,	247.4
3	Top,	608.	3	Top.	451.7
4	Bottom,	366.2	4	Bottom,	203.6
5	Side,	393.9	5	Side,	252.9
6	Side,	372.5	6	Side,	235.9
7	Fin,	423.6	7	Fin,	351.4

In order to get a sample of a carload lot of converter copper it is necessary to take a sample of each charge and mark the number of bars on the ticket that is to go with it. The weight of each charge separately would be the correct way, but as the bars will average about the same, this can be used instead. A number of these charges are bunched together to make a carload, and the samples are cut up into small pieces and as many grammes taken from each as there were bars in the charge. These weighed portions are put together in a large clay or graphite crucible, melted in a blacksmith's forge, and granulated by pouring slowly on to a board set obliquely in a bucket of water. The stream of copper should strike the board about six inches above the water, and should fall a distance of about two feet before striking. It will glance off in fine shots and, chilled by the water, will be found as bright granules when the water is poured off.

CHAPTER XIV.

Design of Converter Plants.

When copper converting was first started in this country the converters were made stationary, and in order to reline them they were filled with water, as described for the old plant at Anaconda. But the experience gained showed that this practice would have to be abandoned. As it was impracticable to wait for them to cool in the stands, it became necessary to remove them and place a fresh vessel in place so that work could go forward without serious interruption.

The Parrot plant was constructed with converters 60 inches diameter by 8 feet 6 inches high, which were removed on a car. The lifting device was four jackscrews, one at each corner of the car, and the heads of these screws impinged on lugs riveted to the converter. The converter was turned on its back by the hydraulic cylinder, the car run under the converter, and the jackscrews applied until the weight of the converter was lifted off the trunnions. The trunnions were then uncoupled at the flange joint, one on each side between the vessel and the stand, the tips of the bearings remaining in the stand. The car bearing the converter was then run out to the relining house and another car with a converter lined and dried was run in place, and after coupling the trunnion tips was ready to receive a charge. This method was discussed for the Anaconda plant, but it was fortunately decided that a traveling crane would be better, and the plant was built according to the plans reproduced in the appendix.

One of the difficulties encountered in this plan was to get the converter and cupolas to deliver their smoke into the same

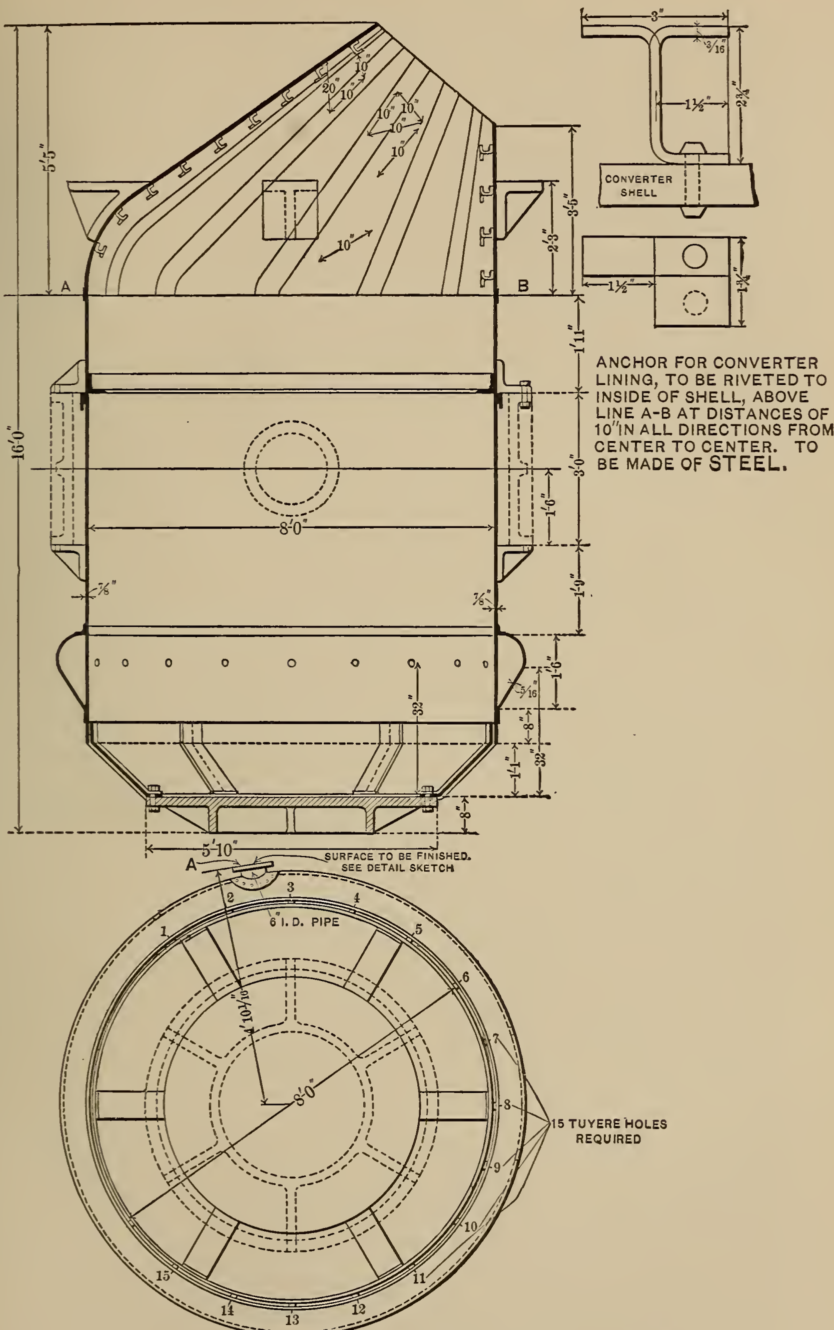


FIG. 20.—COPPER CONVERTERS AT AGUAS CALIENTES.

flue. It had previously been the practice to have the converters turn down towards the cupola to receive the charge and turn up and blow away from it. This would have been impossible with a traveling crane, and the difficulty was overcome by putting two converters to each cupola and making them turn down, away from the cupola, to receive the charge and blow towards it into the same flue.

The first section of the spout from the cupola well was straight and emptied into a broad section which was movable and divided into two spouts, one running to the right and another to the left-hand converter. These spouts were mounted on wheels and could be swung around to either converter when a charge was needed. By means of a few shovels of clay the matte could be diverted to either spout, which would carry it to the converter. The trunnion coupling which was selected for use after many designs had been made is worth special consideration, since it is very strong in construction and can be coupled up or uncoupled in about 30 seconds. The trunnion ring remains on the converter and is taken out with it. The tips of the trunnion ring are conical in shape, with the large ends farthest from the vessel. These cones fit into pockets in the trunnion tips, and are fastened to them by heavy gibs and keys, as well as bolts on the bottom (see Plate V, appendix).

The dimensions of the vessels adopted were 72 inches diameter by 10 feet high. It would probably have been better had they been made 6 feet 6 inches by 12 feet high, but at that time the large converters at Great Falls were not giving satisfaction, although later they improved wonderfully under different management. The discouraging reports circulated had the effect of placing a limit on the size of the vessels at Anaconda.

Since that time it has been the experience of the writer that very large converters do not do as good work on high-grade matte as smaller ones, while on low-grade mattes they do much better. The extremes of size thus far in use are: Parrot, 58 inches diameter by 8 feet 6 inches high; Aguas Calientes, 96 inches diameter by 16 feet high. The converters at Aguas Calientes are illustrated in detail in Figs. 20 and 21. The general arrangement is shown in Figs. 22*a* and 22*b*.

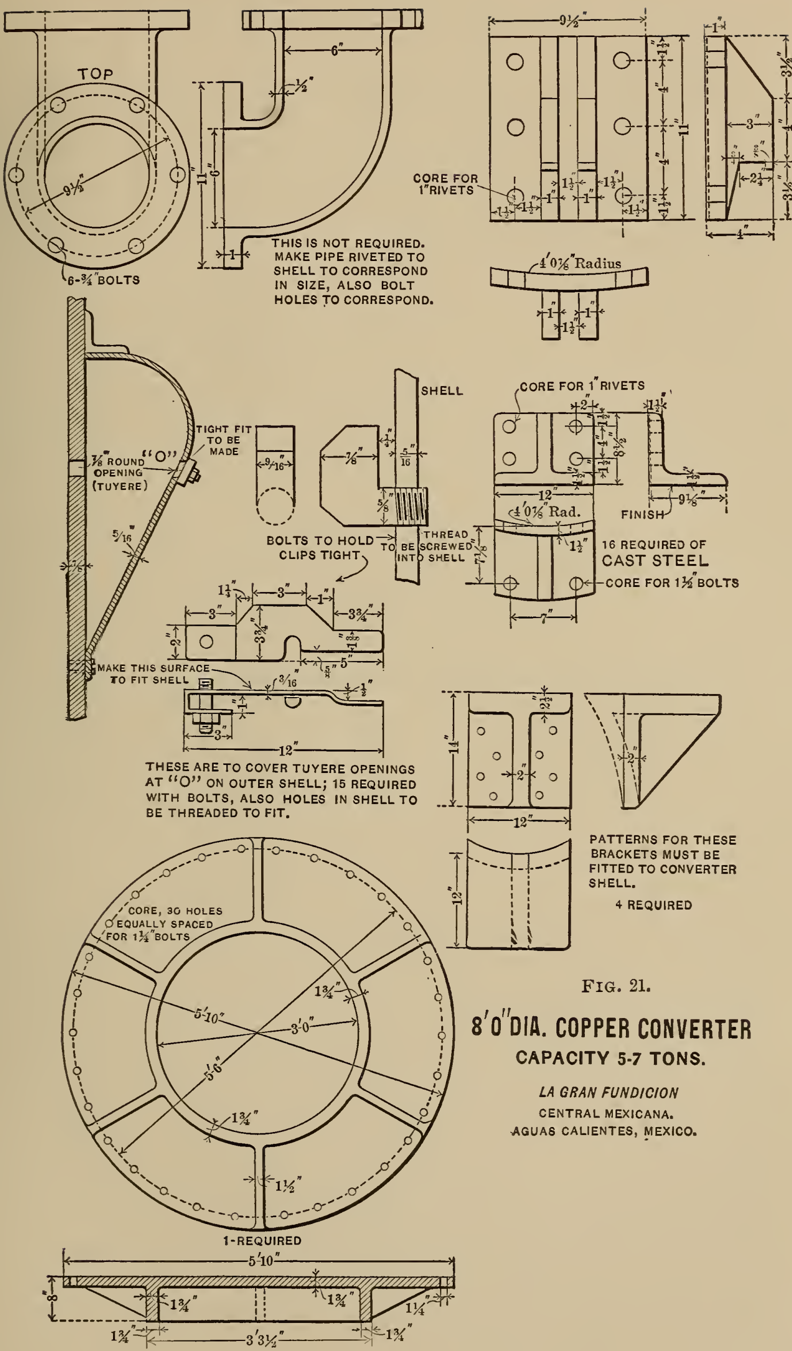


FIG. 21.

8'0" DIA. COPPER CONVERTER

CAPACITY 5-7 TONS.

LA GRAN FUNDICION

CENTRAL MEXICANA.

AGUAS CALIENTES, MEXICO.

The main objection to large converters on high-grade matte is that the slag is very frequently granulated and cannot be poured off, while with the smaller converters, as at Anaconda, no such difficulty was ever experienced; in fact, the trouble was to get the matte high enough in grade so that the linings would last six or seven charges.

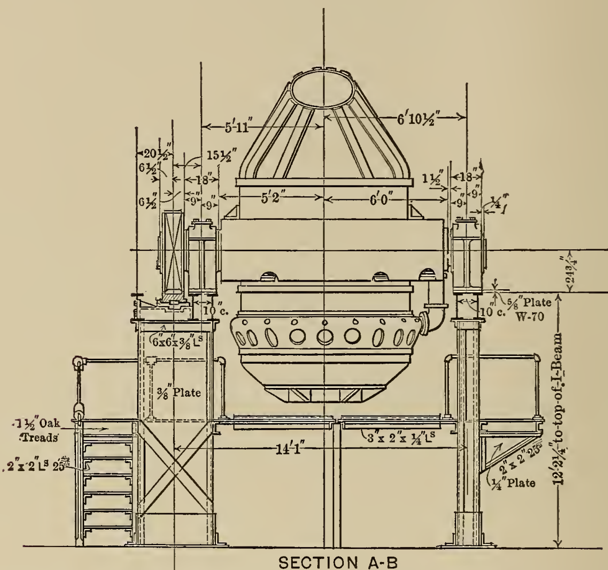


Fig. 22*a*.—General Arrangement of 8-foot Diameter Copper Converter.

As the size of the vessel increases, the thickness of the lining also increases rapidly, and since it is very porous the loss of blast through the lining also increases. A point would soon be reached where with increased thickness of lining the decreased efficiency of the blast would more than counterbalance all the benefits of large vessels. The loss of blast through the lining depends a great deal on how the latter is put in. If it is tamped in around a cone the loss will be less than if it is simply built up by hand. But tamping linings around a cone is much slower and more expensive, and tests made on the life of such linings did not show that they lasted any longer than those put in by simply pounding the wet material into place with the hands.

There are two methods of arranging a converter plant,

either of which is good, and the choice between them depends upon local conditions. If the tonnage is large and a great quantity is to be turned out, the crane plant is by far the better, and the size of the vessels should not be less than 7 feet in diameter by 13 feet high. If the tonnage is small the plant should be constructed on about the same lines as the

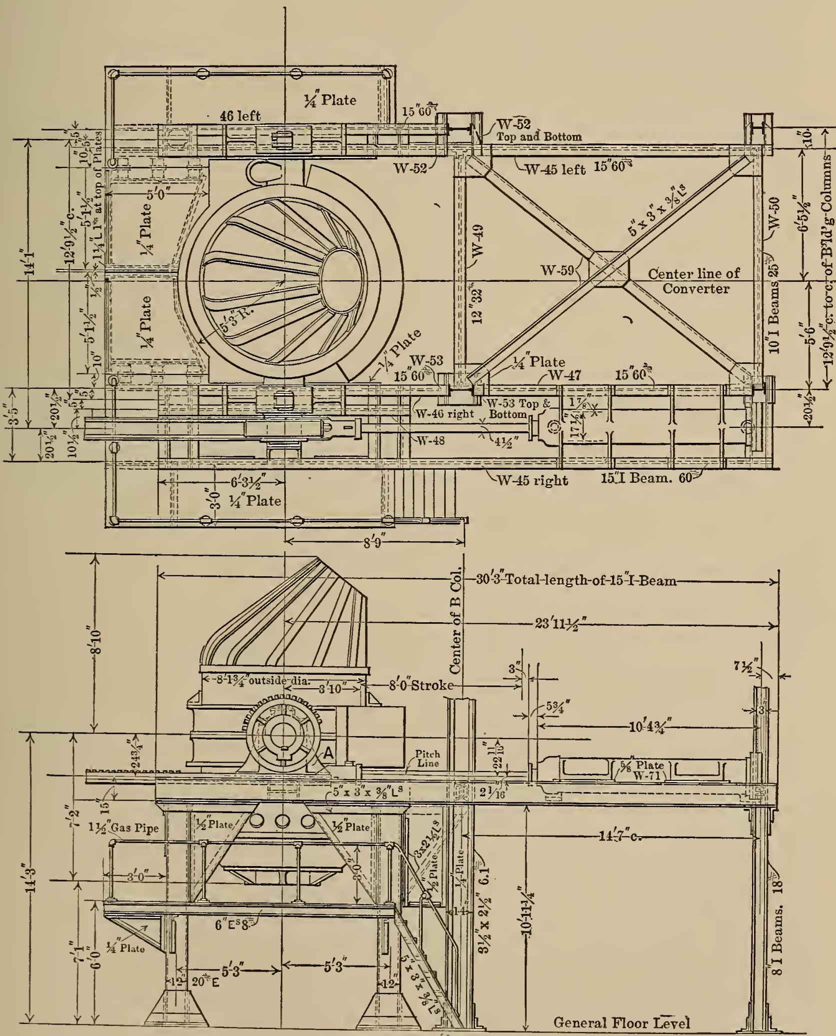
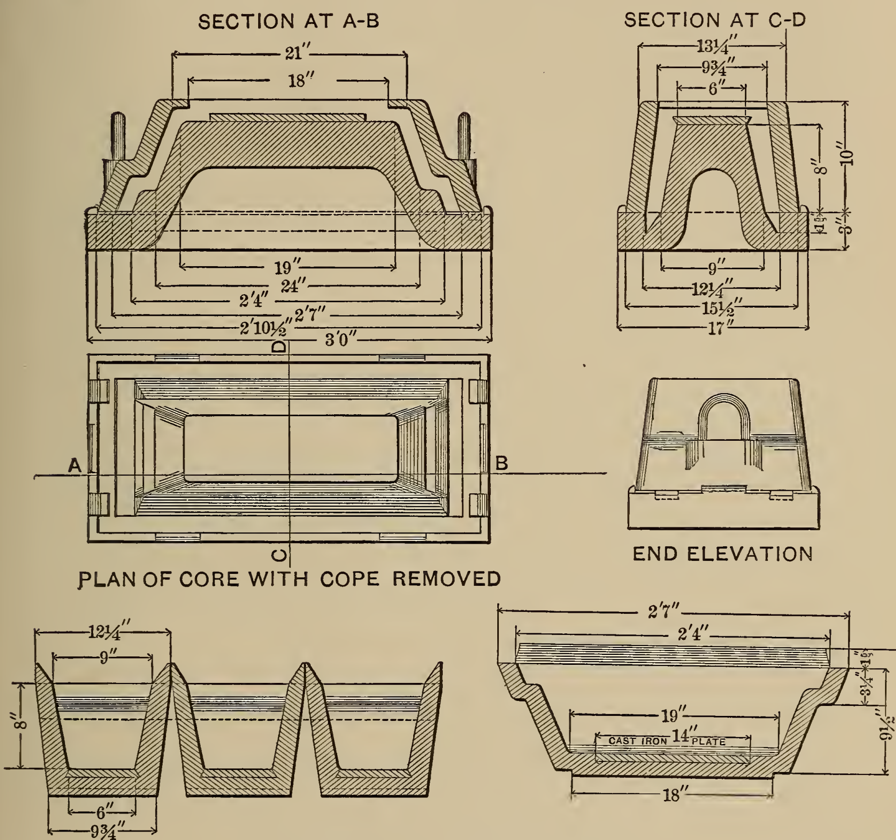


FIG. 22b.—GENERAL ARRANGEMENT OF 8-FOOT DIAMETER COPPER CONVERTER.

Aguas Calientes plant, but with the joint in the lining at the top of the trunnion ring instead of at the bottom. Of the crane plant nothing in addition to what has been said of the Anaconda is necessary, but of the handling by cars at Aguas Calientes this can be said : that the plant will cost much less to construct and a much larger converter can be used than could be handled by any crane that a converter plant could afford to erect. The vessels at this plant, when freshly lined, would weigh fully 40 tons, and the buildings and crane necessary to handle such a great weight would cost fully $30,000 more than the plan adopted.

The practice at this plant was similar to that of most steel plants where the vessels and bottoms are removed on a car, the car with the vessel on it being lifted into position by a hydraulic piston about 15 inches in diameter, acted upon by water pressure of about 500 to 600 pounds to the square inch. The top end of the piston is fitted into a very strong frame, which carries a section of the track long enough to allow the wheel-base of the cars to be moved forward or backward a few inches without running off the end of the rail. The converter is put into place in two sections. The top section being cleaned of slag about the nose, is then taken by the car on to the elevator below the trunnion ring, hoisted into place and bolted to the ring in an inverted position. It is then turned upright by the hydraulic cylinder attached to the trunnion ring and the bottom section fully lined and dried, run under and hoisted into position and bolted to the trunnion ring as well as to the top section. The remainder of the lining must then be put into the nose, and as this is a very long, tedious job the less of it there is to do the sooner it will be done. The mistake was made of cutting the converter in two below the trunnion ring instead of above or at the upper edge. The bottom section should be the largest piece of the shell, so that as much lining as possible may be put in before the top is put on. The idea in constructing the plant in this way was that the bottoms were the only portion of the lining that were corroded, and that they could be removed and renewed just as is done in the steel business. This idea, of course, is wrong. The lining is corroded wherever the slag and matte touch it, except right at the nose, where the slag that is thrown out freezes

and causes it to grow smaller. Owing to the unfortunate circumstance that a lining $2\frac{1}{2}$ feet thick, of clay and quartz, with nothing to support it, will fall out, the top section could not be lined bottom side up and then turned upright to receive the bottom without losing all the lining in the upper section



SECTIONS OF COPPER MOULDS.

FIG. 23.—COPE AND CORE APPARATUS FOR MAKING COPPER MOULDS.

and causing a wreck that would take several hours to repair. If, as stated, the joint had been made at the upper side of the trunnion ring instead of at the lower, the lower section would have been increased by that amount and the quantity of lining necessary to be put in after the cap had been put on greatly reduced. Owing to the length of time necessary to

pass 15,000 pounds of lining in at the nose and put it into place, as well as the time required to dry the lining and heat the vessel to a point where it will be in a fit condition to receive a charge, the time in which a change of vessels could be made was unusually long. From ten to fifteen hours were required as against five minutes at Anaconda from the time a charge was poured until another could be run in. If the joint had been in the proper place a change could have been made in about four hours, including firing the fresh lining.

In other respects the plant was a good one, and the large vessel was certainly a great improvement over a smaller one for the treatment of low-grade mattes. It also made it possible to use for lining material silicious ores, on which there was a margin of profit, which would have been totally unfit for use in smaller vessels.

The use of cast iron or steel moulds for converted copper is rather expensive, and it has been the practice at many places to make the moulds of copper. This is done by pouring refined copper into a mould made of two ells clamped together, and then plunging a core-bar into the metal bath, which would make the mould. It involves considerable expense for labor and supplies, and the experiment was tried of making the moulds direct from the converter. The moulds could be made very easily by means of the cope and core apparatus shown in Fig. 23, but when in use the stream of copper from the converter would strike on one part of the bottom until it became hot enough to weld, and the result would be that the bar and mould could not be separated on account of a union covering a space of about 1½ inches in diameter. If the copper being poured was very hot it would bore holes into the copper moulds as quickly as hot water poured upon ice, and even after the difficulty with the bottoms of the moulds had been remedied by means of the cast-iron plate placed on top of the core before casting, great care had to be taken that the stream did not impinge on the sides of the moulds.

The cope and core were placed on top of an ordinary slag pot, preferably one the bowl of which was cracked, so that it could remain there without being changed. The cast plate shown was then placed on top of the core and the pot run under the converter nose. The copper was poured in slowly to allow

the SO_2 gas to escape and the space between the cope and core to fill up solidly. The pot was then pulled to one side and the cap lifted off by means of the traveling crane, and a bar with chisel point driven between the copper mould and the cast core at one end. A couple of men would then pry down on the bar until the mould would come away from the core, or until the mould and core were raised from one side of the pot, when a few sharp blows from a sledge on the raised end of the core would bring them apart. The cast iron would come away with the copper mould and would serve the double purpose of protecting the core while making the moulds and preventing the welding of bars to the moulds while in use.

This scheme worked very satisfactorily, and copper moulds with cast-iron bottoms were used entirely. It was necessary to make from 12 to 20 each day owing to their breaking after about five to seven days' use. The scrap or broken ones were, of course, available as bars for the casting furnaces.

The life of these moulds depended a great deal on the kind of finish copper that was used in making them, moulds made from blister finish being found much better than gas finish. While making moulds two or three cores on pots would be used at one time, one filling while the others would be stripping or being washed in limewater to prevent them being burned. By the use of a few extra men and a traveling crane twenty moulds could be made in an hour.

CHAPTER XV.

LINING A CONVERTER.

It is unfortunate that in most places the quartz and clay are so entirely distinct. If, as at Aguas Calientes, they are both contained in one ore the lining is much more homogeneous and compact. The linings in Montana are made from quartz which has no plastic or adhesive qualities and is very refractory, and a fat, sticky clay, which is not refractory and which melts away from between the particles of quartz, allowing them to drop off and mix mechanically with the slag or copper. This probably accounts for much of the difference in the composition of the converter slag at Anaconda and Aguas Calientes, which averaged as follows:

	SiO_2.	FeO.	Cu.
Aguas Calientes	25 per cent.	62 per cent.	5 per cent.
Anaconda	36 per cent.	49 per cent.	5 per cent.

The matte at Anaconda would average 55 per cent. and at Aguas Calientes not more than 35, and part of the difference in slag composition is due to the fact that high-grade matte makes more silicious slags than low grade.

The introduction of silica through the tuyeres was not attempted at Anaconda, for the reason that it was apparent to the writer that so much silica as would be required could not be blown in within the short time allowed, and, second, because it would act as a sand blast and ruinously cut into the ironwork of the converter in a very short time.

Even if the silica could be introduced in this way it would still produce another trouble in the converter. The small particles of silica being cold would combine very imperfectly with the iron oxide and produce granulated and pasty slags, which could not be skimmed or poured out of the vessel.

About 60 tons of quartz and 7 tons of clay were consumed daily for lining, and two sets of crushers and rolls and three Chilian mills were kept going day and night to grind the material for lining. The mixture used in charging the Chilian mills was 40 shovels of quartz to five of clay. This was ground together with hot water in winter time, and cold when the weather would permit, until the whole mass was reduced to the consistency of a stiff mud and the quartz pulverized to the size of peas. In this condition it was discharged from the mills into a stock-pile or direct into wheelbarrows to be taken to the converters in process of lining. Before passing to the liner it was pounded with the shovel until it adhered thoroughly together, and was then cut into cubes by the shovel and passed into the converter. As soon as the converters were removed from the stands the nose section was taken off by removing the keys from the key bolts and lifting with the crane, when the joint would generally break, but if it did not, a wedge would be driven in and a fracture started. To hasten the cooling the interior would be sprinkled with water from a hose. The skin of slag was then cut out and the bottom section relined, after which the top was put on by the crane and the lining finished as high above the joint as possible. The converter was then taken by the crane to a place where there was a blast connection through a three-inch hose to a branch from the main supplying the blast furnace. A fire was kindled with oily waste and dry wood, and shortly afterwards coke was put in and the blast turned on. The fire was kept up until the converter was required for use in the stands. From three to four hours was required to dry and bake the lining sufficiently to insure it from falling out when the vessel was inverted. It was found that if the fire was allowed to go out the lining would contract so much that the probabilities were it would collapse when put in the stand and turned upside down.

The life of a lining should not be measured so much by the time or charges as by the copper produced and the matte converted. The production of a lining is dependent on the composition of the clay and quartz, as well as upon the grade of matte converted. If the clay is not plastic or the mixture poorly ground, the lining may collapse after a single charge.

On the other hand, if all things are working to the best advantage and the grade of the matte will average 55 per cent. Cu, the first charge for a six-foot vessel should produce about eight to twelve bars of copper of about 250 pounds to the bar; the second charge from ten to sixteen bars, the third from sixteen to twenty, and the fourth from eighteen to twenty-eight, the fifth from twenty to thirty, the sixth from twenty to thirty-five, and so on until the lining becomes too thin to stand another charge. A production of 100 bars is a good run for a lining on 55 per cent. matte, and all statements to the contrary must be taken with considerable doubt. In one work on the subject a statement is made that the linings are usually exhausted after the ninth charge. It is safe to say that with the size of vessels in use at the Parrot works, where this remarkable work was done, all the linings that have made nine charges in the past two years can be counted on the fingers of one hand. At Aguas Calientes, where the vessel was 8 feet in diameter and the lining 2½ feet thick at the tuyeres, one charge of 30 per cent. matte weighing 40,000 pounds would, if added 8,000 to 15,000 pounds at a time, finish about forty to forty-five bars and corrode the lining so much that a second charge could not be finished. With such low-grade matte the second charge would only be blown to white metal and skimmed, the 80 per cent. copper matte being poured into beds and returned to the next charge as scrap. A single lining would, including the washout, convert about 50 tons of 50 per cent. matte.

The experiment was tried at Anaconda of turning the converter on its back in the stand and putting in a large patch of green lining wherever needed, and allowing it to dry thoroughly before a charge was run in. By repeating this operation several times twelve charges were finished with a total production of 212 bars for a converter 60 by 60 inches square. Aside from the large vessels at Aguas Calientes this is the largest production for a single lining that I know of. The largest charges thus far finished were probably made at Great Falls, where according to report something over 80 bars, or about 16,000 pounds, of copper have been poured. The largest single charge at Aguas Calientes was 75 bars.

CHAPTER XVI.

CASTING ANODES DIRECT FROM CONVERTER.

The casting of anodes from the converter had been attempted at the old plant at Anaconda, and was successful enough to indicate that it could be done by the assistance of a traveling crane. When the new plant was well started and the men had become accustomed to the use of new appliances, the casting of anodes was carried on for some time, although eventually abandoned because of the objection made that the impurities in the converter anodes caused the electrolyte of the refinery to become too impure, it being stated that the cathode copper from such anodes would be low in conductivity and unfit for wire bars. However, it was demonstrated that anodes of reasonably uniform weight and density could be cast at a saving of about 0.35 cent per pound, or $7 to the ton. The anodes were cast on edge between cast-iron moulds grouped together so that the face of one mould would be the back of the next, the set being held together with iron clamps with springs to allow for expansion when filled with copper. Owing to the rapid chilling of the copper when it was poured into the moulds, it was necessary to be able to pour along the entire length of the mouth of the mould, as well as to change quickly from one mould to the other and back again. The escaping SO_2 gas would cause the copper to shrink in the moulds, and the anodes would be hollow unless filled a second time. To overcome these difficulties a car with a movable table-top on differential rollers was designed, and the set of moulds put on this table. All the peculiar difficulties were overcome except the warping of the cast-iron moulds by the heat from the copper. This caused the anodes to be somewhat irregular in thickness and weight, owing to the fact that the moulds after being used for a few days would buckle so much

that they could no longer be drawn close together by the clamp. This variation would amount to 20 per cent. of the weight of the anodes, in moulds made to cast plates of equal thickness. The anodes were taken from the moulds to a large Gate shear, where the ragged upper edge was sheared off, and two holes punched in the ears by which they were suspended in the tanks.

It was found that plates could not be cast less than 1 inch thick on account of the chilling of the copper before the mould had been filled in all parts. Anodes $1\frac{1}{8}$ inches in thickness of the dimensions required would weigh 230 pounds made of converter copper, and if made of cast copper would weigh considerably over 300 pounds, the difference in weight being due to the porous character of the converter copper caused by the escaping SO_2 gas. Anodes of unpoled and converter copper, which contain more impurities than were contained in these, are used at other refineries, but with what success as regards the conductivity of the cathode copper I am unable to state.

The average assay of samples of converter copper showed 99 per cent. Cu, with silver varying from 80 to 120 ounces and gold from two-tenths to five-tenths ounce to the ton. There was considerable SO_2 gas retained in the pores of the anodes, which it was stated was converted into sulphuric acid in the electrolyte, resulting in the increase of acidity of the solutions and rendering unnecessary the addition of free acid. This, as well as the porous character of the converter anodes which later would present greater surface to the action of solutions, should be considered as points in their favor.

If the anodes are cast from the converter in open moulds, as at Great Falls, the thickness and weight are subject to greater variation than if cast on edge. The stream of copper, as it comes from the converter, has to be broken up and deflected to a different part of the mould by allowing it to fall on a board held by an attendant. If this is not done the copper will set before the lugs have been made, and the anode will be much thicker in the middle than on the edges.

There is a much greater production of scrap in the refinery from converter anodes than from cast anodes, owing to the imperfections of the former.

CHAPTER XVII.

Cost of Producing Copper at Anaconda.

In explanation of the following figures, in case they should not seem clear and intelligible, it is only necessary to state that the losses in percentage are on a basis of the material charged to the different departments. To get the copper marketed it is necessary to deduct them in their order from 100 per cent. in the ore after multiplying by the percentage delivered to that department.

For example, if 18 per cent. is lost in dressing and 9 per cent. in smelting, then 100 — 18 = 82 per cent. delivered to smelter; 82 × 9 per cent. = 7.38 per cent. of Cu in the ore lost in smelting; 82 per cent. — 7.4 = 74.6 delivered to converter; 74.6 × 3 = 2.238 per cent. of Cu in ore lost in converting; 74.6 — 2.2 = 72.4 delivered to casting; 72.4 × 1 = 0.72, and 72.4 — .7 = 71.7 delivered to refinery, etc.

In the same way the costs are figured, only the cost for the department is divided by the per cent. of Cu marketed. Starting at 100 per cent. and working backwards, 1 per cent. loss in melting would have the cost divided by 99 per cent., 1 per cent. loss in casting by 98, etc.

Copper in ore	100 per cent.
Loss in dressing	18
Delivered to smelter	82
Loss in smelting	9
$82 \times .09 = 7.38$ (say 7.4); $82 - 7.4 = 74.6$	
Copper delivered to converter	74.6
Loss in converting	3.0
$74.6 \times .03 = 2.238$ (say 2.2); $74.6 - 2.2 = 72.4$	
Copper delivered to casting department	72.4
Loss in casting	2.0
$72.4 \times .01 = 0.724$ (say 0.7); $72.4 - 0.7 = 71.7$	
Copper delivered to refinery	71.7
Loss in refining	0.5
$72.4 \times .005 = 0.3585$ (say 0.3); $71.7 - 0.3 = 71.4$	

Copper delivered to melting department	71.4
Loss in melting	1.0
$71.4 \times 0.01 = 0.714$ (say 0.7): $71.4 - 0.7 = 70.7$	
Copper finally recovered from ore	70.7
Cost of dressing 0.53 per pound of copper in concentrates.	

Making the calculation in the same manner on the basis of the copper contents of the concentrates delivered from the dressing works, it appears that 86.2 per cent. is recovered, *i. e.*, the loss in smelting and converting is 13.8 per cent. The cost of dressing per pound of copper marketed is consequently: $0.53 \div 0.862 = 0.614$. In a similar manner the cost of smelting per pound of copper marketed works out: $2.035 \div 0.945 = 2.153$; cost of converting matte to blister: $0.6870 \div 97.5 = 0.705$; cost of casting: $0.35 \div 98.5 = 0.356$; cost of refining, 1.00 cent; cost of melting, 0.40 cent, cost of mining: $2.2 \div 70.7 = 3.112$. The recapitulation is as follows:

Cost mining per pound Cu sold	3.112 cts.
Cost concentrating per pound Cu sold	.614
Cost smelting per pound Cu sold	2.153
Cost converting per pound Cu sold	.705
	6.584
Cost casting per pound Cu sold	.356
Cost refining per pound Cu sold	1.000
Cost melting per pound Cu sold	.400
Total cost per pound Cu sold	8.340

To each pound of copper there is recovered an average value of about 4 cents in precious metals.

The building at Anaconda was designed for twelve converters running and thirty-six shells, being three shells to each stand. There were to be six cupolas, one to every two stands, and two blast furnaces to work over the converter slag. Three No. 7 Roots blowers were required to furnish the blast for the cupolas, blast furnaces, and drying out the converters.

There were four blowing engines with a capacity of 2,500 cubic feet each per minute and two with a capacity of 8000 cubic feet each, making a total capacity of 26,000 cubic feet per minute, or about 2,200 cubic feet per minute for each converter in operation.

To furnish the steam there were eight firebox boilers with shells 72 inches by 18 feet besides the firebox. Each had a steaming capacity of 160 indicated horse-power, making 1,280

horse-power in order to turn out 11,000,000 pounds of copper per month. This would be the maximum if all were running at one time, but out of twelve converters seldom more than eight were running, and as the speed of the engines was regulated automatically by the air pressure, the maximum power was seldom required. The blowing engines were all furnished with Corliss steam valves, and the four small ones with Corliss air valves. The large engines were built according to specifications with gridiron slide valves for the air cylinders, and it was found that these were much better than the Corliss valves. The cost of this plant was about $400,000.

APPENDIX.

SPECIFICATIONS OF BUILDINGS AND MACHINERY FOR COPPER CONVERTER PLANT, ANACONDA MINING COMPANY, ANACONDA, MONTANA.

All the machinery described in the following specifications is to be first-class in every respect, both as regards material and workmanship. The machinery described is to be complete as per specifications and blue prints furnished, omissions in specifications notwithstanding.

BUILDINGS: General appearance and dimensions as per Plates I and II.

Note: Dimensions given are from *C* to *C* of posts.

STOCKHOUSE: 324 ft. long, 23 ft. wide, and supported on posts 12 ft. *C*; on one side these posts are 18 ft. high; on the other side they are 35 ft. high. The 18-ft. posts are attached to and stand on 24-in. deep plate girders; these girders are securely fastened to the 35-ft. posts, and are to be provided with angle irons or knees riveted to top of girders a suitable distance apart to fasten 12 x 14-in. stringers to for a standard track (4 ft. 8 in. gauge); track to be in center of building. Roof to have a longitudinal opening 4 ft. wide the whole length; the upper side to be provided with window openings—two windows, twelve 10 x 18-in. lights in each bent. The lower side towards the Converter Building to be left open.

CUPOLA AND CONVERTER BUILDING: 240 x 70 x 26 in. Trusses are attached to the 35-ft. posts of the above-mentioned building in the rear, and in front on 26-ft. high posts 12 ft. *C*. These posts, or a suitable number of them, are provided with brackets to carry a girder for traveling cranes. Roof to be covered, except the nine ventilator holes 12 x 17 ft. 6 in., and the holes for six cupola stacks 12 x 8 ft.; a frame to be

made around respective openings and their sides to be covered, and a suitable flashing made between sides and roof. The ends of this building to be covered on a suitable framework to within 8 ft. from the ground, except where adjoined by buildings.

Blast Furnace Building: 84 x 36 x 29 ft. high. Trusses are attached to posts of Stockhouse Building in the rear and on 29-ft. posts 12 ft. *C* in the front. Roof to have three openings 8 x 12 ft. for furnace stacks, located as per drawings. Roof and sides to be covered within 8 ft. of the ground.

Silica Mill Building: 72 x 56 x 20 ft. Trusses are supported on one side on a stone wall; wall plates and anchor bolts for this. On the other side they are supported on 20-ft. high posts 14 ft. *C*. One end and one side are provided with window openings, as per drawing. Roof has two dormers 14 ft. wide 8 ft. high, located as per plan. Roof and sides to be covered except for doors in the end adjoining buildings.

Shipping Platform, in front of Converter Building, is covered with a roof 240 x 31 ft. Trusses supported on 14-ft. posts 12 ft. *C*, and attached to Converter Building posts. Sides are left open 8 ft. from ground.

All buildings to be iron constructions, well braced, and executed in a good and workmanlike manner. All roofs to be covered with 2½ in. corrugated iron No. 20, and sides, as specified, with 5½ in. corrugated iron No. 22. All to be painted with good mineral paint.

SPECIFICATIONS OF MACHINERY FOR NEW COPPER CONVERTER PLANT FOR ANACONDA MINING COMPANY, AT ANACONDA, MONTANA.

To consist of

36 Converters.
12 Sets of stands and trunnions for same.
12 Hydraulic cylinders, rack, gears and valves.
6 Copper cupolas, and 3 extra wells.
3 Blast furnaces, 6 forehearths.
12 Double mould cars.
12 Sets of runners.

12 Steel-plate flues from converters to main flue.
2 Sheet-steel blast pipes.
2 12-ton traveling cranes.
3 No. 7 Roots blowers.
2 Chilian mills.
12 Matte carts.
100 Slag pots.
6 Colorado Iron Works dumping slag-trucks.

MACHINERY.

Plate II shows the general plan and location of machinery in buildings.

CONVERTERS: The converters are to be constructed as per Plates III, IV, V, and VI; to be made in two sections. The lower section 6 ft. diameter by 5 ft. high; to be made of 7-16 tank steel. To it is riveted a dished and flanged head, also made of 7-16 steel. In the center of this head is a 1-in. hole for drainage of moisture from lining. The nose or upper section to be made of $\frac{3}{8}$-in. tank steel. Half way around the 24-in. pouring opening is riveted a $\frac{3}{8}$ x 12 in. wide reinforcement plate. Both sections are reinforced by 3 x 3 x $\frac{1}{2}$-in. angle iron rings, securely riveted to them. A band 7-16 x 10 in. wide is riveted to lower section where lugs are attached.

LUGS: Eight wrought iron or forged steel lugs to be riveted to each section located respectively as per drawings. Four of the lugs on nose section to have a loop or eye made in their upper ends for attaching chain hooks for lifting converters. Another eye for lifting the nose section alone is riveted on top, and should be so located that this section will hang level when lifted.

GUIDES: To the nose section are further riveted four straps 4 x $\frac{3}{4}$-in. iron, projecting 2 in., overlapping the lower section, so as to guide and hold sections in central positions when put together. All rivets used in construction of shells to be 11-16 in., Burden brand, 3 in. pitch.

LINING ANCHORS to be made of 3-16 x 2-in. steel (see Plate III) and riveted to the interior of nose section and bottom of lower section, spaced about 10 in. *C*, 9-16-in. rivets.

Tuyere Boxes: Eight tuyere boxes to be riveted to shell. Dimensions and location as per Plate VI. Eight holes, corresponding to tuyere holes in boxes, to be cut in converter shells, and the edges of said holes to be calked and made air-tight.

Tuyeres are to be made of cast iron, as per drawing.

Tuyere-Box Covers to be fitted with two valves, as shown in details, Plate VI, for facilitating the punching of tuyeres.

Bustle Pipe: A rectangular cast-iron box 4 x 7 in. inside dimensions, to be made in three sections, provided with eight short nozzles corresponding to receptacles in tuyere boxes, and secured with two $\frac{3}{4}$-in. bolts through each box, making an air-tight joint, with suitable packing.

Trunnion Ring: Cast-iron trunnion rings made in two sections; joints to be faced and bolted together with eight $1\frac{1}{2}$-in. turned bolts, fitted in reamed holes. Trunnion tips to be turned to gauge and templets in order to be interchangeable. Trunnion rings are provided with eight sleeve holes for 2-in. key bolts; bolts to be made of Norway iron, and keys of steel. The bolt holes in above-mentioned lugs to correspond exactly with sleeve holes in trunnion rings.

Blast Connection between trunnion ring and tuyere box to be made with 5-in. pipe and fittings.

Journals to be turned, bored, and planed as indicated on drawing, and made to fit trunnion ring and tips perfectly. A wrought-iron strap to be fitted to box part of trunnion where jib and key pass through, as shown in detail drawing.

Boxes to be bored to fit trunnion iron to iron and planed in joint and underneath.

Stands (Plates VII and VIII) to be planed on top for boxes and also on projection or washers for reach rods for hydraulic cylinder where indicated on drawing.

Gear: Cast-iron gear, neat fit on trunnion, and secured by two keys; shrouds to be turned; diameter to be equal to pitch-line.

Note: Referring to Plate II, general plan of machinery, it will be seen that it will be necessary to make portions of stands and other parts right and left.

Hydraulic Cylinder: Plate IX is a detail drawing of hydraulic cylinder, rack and gear, etc. Cylinder, 15-in. bore,

7 ft. ½ in. long (6 ft. 4½-in. stroke) 1⅛ in. thick; cylinder to be bored true, and piston to be fitted with two brass rings bored and turned concentric, rubber or square fiber packing to be used between piston and rings; ground joint to be made between follower and piston; follower to have two ¾-in. tap-holes on top for eye bolts. Cylinder heads to have spigot joints and be bolted to cylinder with twelve 1-in. turned bolts. Piston rod, 4-in. steel, 9 ft. 6½ in. long. Cylinder to be counter-bored, so piston will overrun and leave no shoulders.

RACK: 9 ft. 1 in. long, to be planed on shrouds and back where roller rests attached to piston rod by socket and key. A keyhole is also made above rod, in order to back out rod when necessary. Cylinder is attached to stands by four 2½-in. rods. Four pieces of extra strong 3-in. wrought-iron pipes act as sleeves; these must be faced off exactly to one length.

HYDRAULIC VALVE: Plate X is a detail drawing for a 4-way valve. Slide valve, stuffing box, nut and gland to be brass castings; valve stem steel forging; the teeth on it to be cut out of the solid.

RUNNERS: Plate XI is a detail drawing for runners and their supports; also platform from cupola foundation to runners. One straight runner 10 x 12 in. by 12 ft. 8 in. long, made of 3-16 tank steel and 2 x 2-in. angle irons attached to cupola with a cast-iron socket leads to the double runners, which also are made of 3-16-in. tank steel and 2 x 2 angle irons with curved cast-iron mouthpieces. As shown, they are pivoted on a casting suspended between two 8-in. channel irons. At the extremity of these channel irons are attached hangers for a circular track, on which the supporting rollers rest.

CONVERTER FLUES: Plate XII is a detail drawing of the steel-plate flue leading from converters to main flue. It is made of 3-16-in. tank steel and 2 x 2 angle irons, all but the first two feet nearest converter, which is made of 5-16 tank steel. Flue to be suspended from cupola floor beams, and traveling crane girder by suitable straps.

TRAVELING CRANES: Two 12-ton cranes 42 ft. 3 in. span, required for lifting converters out of and into their bearings. Cranes to be operated by electric motors. Two speeds required for hoisting; one for lifting the whole converter, and

a faster speed for handling the nose section alone. Trolley to be traversed, and cranes to travel the whole length of building. Plate XIII is a cross-section of building, side elevation of crane, and of one girder for traveling crane and position of posts. Stay braces for supporting this girder to be bolted to retaining wall in front of cupolas; the other girder, as mentioned above, to be supported on bracket riveted to building posts.

MOULD CARS, constructed as per drawing of 6-in. channel beams and 3-16-in. tank steel, cast-iron axle boxes (babbitted) and flange wheels. Twelve double cars required.

FLOOR PLATES: Corrugated floor plates, manhole plates, etc., as per drawing,

CUPOLAS: Plate XIV is a detail of cupola; six cupolas required. The well, 5 ft. 4 in. diameter by 4 ft. 6 in. high, is made of 5-16 tank steel, in halves, jointed on the sides by 3 x 3 angle irons, riveted to shell and bolted together.

WATER-JACKET is made tapered, inside diameters being 46 in. and 54 in.; total height 3 ft. 3 in.; to have a water space $4\frac{1}{4}$ in. Inside shell made of $\frac{3}{8}$-in. flange steel, and outside shell 5-16-in. steel, provided with four cast-iron ears for feed water and overflow, located as per drawing; also four hand holes, plates, and crabs; 11-16-in. Burden rivets, $1\frac{3}{4}$ in. pitch, to be used.

CUPOLA SHELL: 66 in. diameter by 6 ft. high, made of 5-16-in. tank steel. Four cast-iron brackets are riveted to it and eight wrought-iron knees; similar knees to be attached to water jacket for hanging same to cupola shell by eight 1-in. rods. To the lower end inside and to the upper end outside of shell are riveted 3 x 3-in. angle iron rings.

STACK: A 36 in. diameter by 24-ft. high stack, made of No. 8 steel, is connected to shell by a 3 ft. long cone, its lower end being flanged and bolted to the above-mentioned angle iron ring. Top of stacks to be provided with cast-iron dampers, suitable levers and fulcrums; a band $\frac{3}{8}$ x 3 in. is riveted to top of stacks. A branch 36 in. diameter leading to main flue made of 3-16 tank steel, provided with a butterfly valve, as shown on Plate XIII.

CHARGING-DOOR opening in cupola is reinforced by a $\frac{1}{2}$ x 2-in. wrought-iron frame, riveted to inside of shell.

Tuyeres: Four tuyere points with peep holes for each cupola. Tuyere points are connected to bustle-pipe nozzles by canvas sacks.

Charging Floor (Plate XV) consists of 5-16-in. corrugated wrought-iron plates on 8-in. steel I-beams supported on wrought-iron columns, provided with cast-iron top and bottom caps. Columns supporting cupola to be cast iron, with brackets for floor beams, as shown on plan.

Blast Furnaces: Plate XVI shows details of furnaces. Three furnaces complete.

Crucible: Crucible plates to be made of cast iron 1½ in. thick, well ribbed and bound together by bolts; they fit in a pan made of 3-16-in. tank steel, with 3 x 3-in. angle iron frame, riveted to its edges; one end to be provided with a cast-iron spout or taphole.

Water-Jacket: To be made in four pieces. Sides to be made of ⅜-in. Park Bros.' flange steel; each side to have six tuyeres 4 x 5 in. rolled and beaded. Inside sheet of end pieces to be ⅜-in. Park Bros.' flange steel; outside sheet 5-16-in. tank steel, riveted together on a 2 x 2-in. angle-iron frame. Water-jacket to be 5½ in. thick, and each side piece to be provided with two cast-iron ears for feed and overflow, and each end piece to be provided with one cast-iron ear. Each piece of water-jacket to have 1¼-in. gas-pipe washout holes and two hand holes, plates and crabs. One end piece to have an arch over cast-iron spout, as shown. 11-16-in. Burden rivets, 1⅝ in. pitch, to be used. Jacket to be well stay-bolted and bound together with key rods, as shown.

Brickwork: Above water-jacket will be built on a cast-iron plate supported on a rectangular frame, consisting of three 8-in. steel I-beams mitered together; an angle iron knee to be riveted in each corner of inside frame. I-beam frame to rest on four cast-iron columns 9 in. diameter 7 in. core, having suitable top and bottom base plates. Brickwork to be bound together by 3 x 3 angle iron on corners and eye-bolt rods as shown.

Door Sills: Cast-iron plates 22 x 48 x 2 in. thick for charging doors, two for each furnace.

Stack Plate: A cast-iron frame 8 ft. 4 in. by 6 ft. 2 in. by 1 in. thick, with a 1½ x ¾-in. rim all around outside edge

and a 2 x $\frac{3}{4}$-in. rim on top around a 6 ft. by 3 ft. 10 in. hole for the 6 ft. 2 in. by 4 ft. stack to fit into.

STACKS to be 48 in. diameter by 20 ft. high, made of No. 8 steel and provided with cast-iron dampers, levers, and fulcrums. A $\frac{5}{8}$ x 3-in. band around top of stack. A 48 in. diameter branch, 3-16 in. steel, leading to main flue provided with a butterfly valve.

CHARGING FLOOR: A 3-in. plank floor on 3 x 12-in. joists are carried on 10-in. steel I-beams supported on retaining wall, intermediary 6-in. wrought-iron columns and attached to building posts. A strip 4 ft. wide by 84 ft. long in front of chute from stockhouse to be covered with 3-16-in. wrought-iron plates, and a space 4 ft. wide by 18 ft. long on each side of furnaces, and 4 x 74 ft. in front of furnaces, to be covered with $\frac{3}{4}$-in. cast-iron plates, as indicated on drawing.

FOREHEARTHS: 4 x 3 ft. 6 in. x 2 ft. high to be made in halves of 5-16-in. tank steel and $2\frac{1}{2}$ angle irons, jointed on the sides by 3 x 3-in. angle irons riveted to box and bolted together. Forehearths to be provided with two cast-iron detachable spouts, located as per drawing.

TUYERE POINTS: Twelve tuyere points with peep holes for each furnace, connected to bustle-pipe nozzles by canvas sacks.

BLAST PIPES: Plate XVII general plan and elevation of blast pipes and details of bustle pipes for cupolas, and blast furnaces and converters.

Note: Connections to blowers and blowing engine receivers with above pipes to be made from actual measurements when in place.

MATERIAL: Blast pipes to be made of black sheet steel; thickness for the different sizes of pipes are marked on drawings.

BLOWERS: Two No. 7 Roots blowers, bottom discharge.

CHILIAN MILLS: Two Chilian mills, of T. Carlins & Sons' make (Allegheny, Pa.), wet grinding pan 7 ft. 6 in. diameter, as described in their catalogue, page 115.

MATTE CARTS: Twelve, as per Plate XVIII.

SLAG POTS: One hundred, as per drawing.

INTENTION OF THESE SPECIFICATIONS.

(1) It is the intention of these specifications to have all converters, furnaces, and all the necessary appliances for a complete plant; to be made of the best material and workmanship. All the castings to be made true to form, to be free from blowholes, cold laps, or scales, or imperfections of any kind. The forgings to be smooth, true to form, and free from defects of all kinds, and the best material must be used in same.

(2) All the several parts of converters, furnaces, etc., must be made interchangeable, and all finished parts to be made to templets.

(3) When the wording of any clause of the specifications may not be clearly understood, or is open to misconstruction, or when the plans are at fault from errors or mistakes, the company is hereby authorized to give the necessary explanation through its engineer, his decision to be final in each case; and all the work contemplated and described by plans and specifications shall be done to the satisfaction of the Anaconda Mining Co., or its engineer, who shall judge as to the fitness of materials used in the construction of their several parts, and he shall have the right of correcting any errors or omissions in the plans or specifications, when such corrections are necessary for the proper fulfillment of their intentions.

(4) Further detailed drawings, if necessary, will be made by the contractors, and approved by the company's engineer, or be furnished by him, as the case may require, in order to carry out the work and intent of the specifications.

(5) These specifications are accompanied by blue prints, marked "Copper Converter Plant for Anaconda Mining Co."

(6) All work must be set up in shop with necessary attachments, and properly marked so as to facilitate erection.

COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
JAN. 14, 1893.

PLATE I

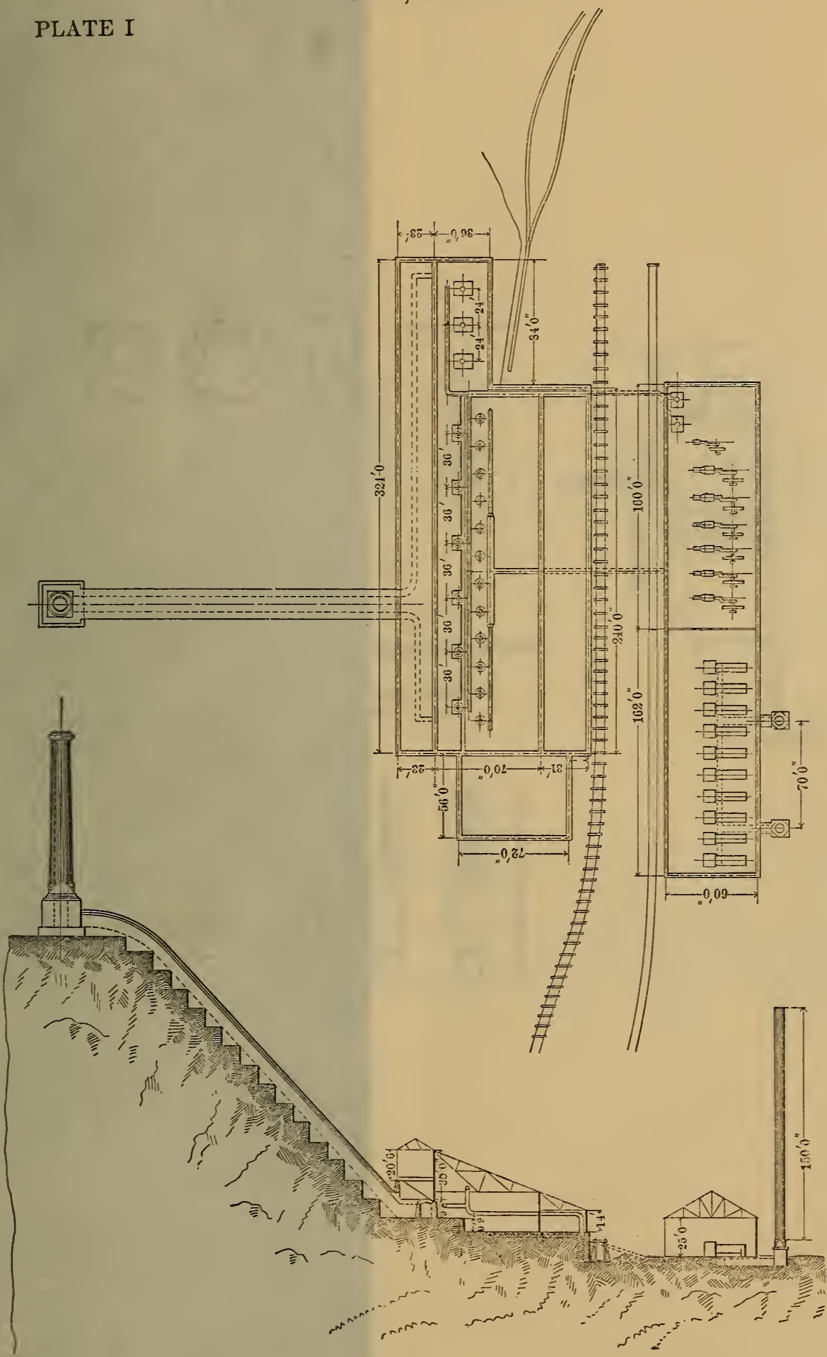
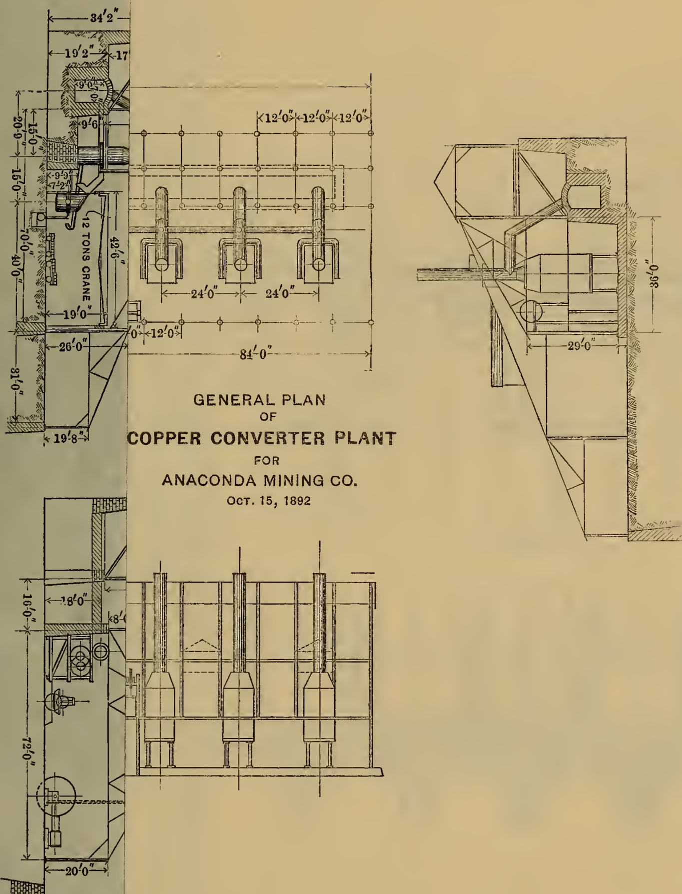
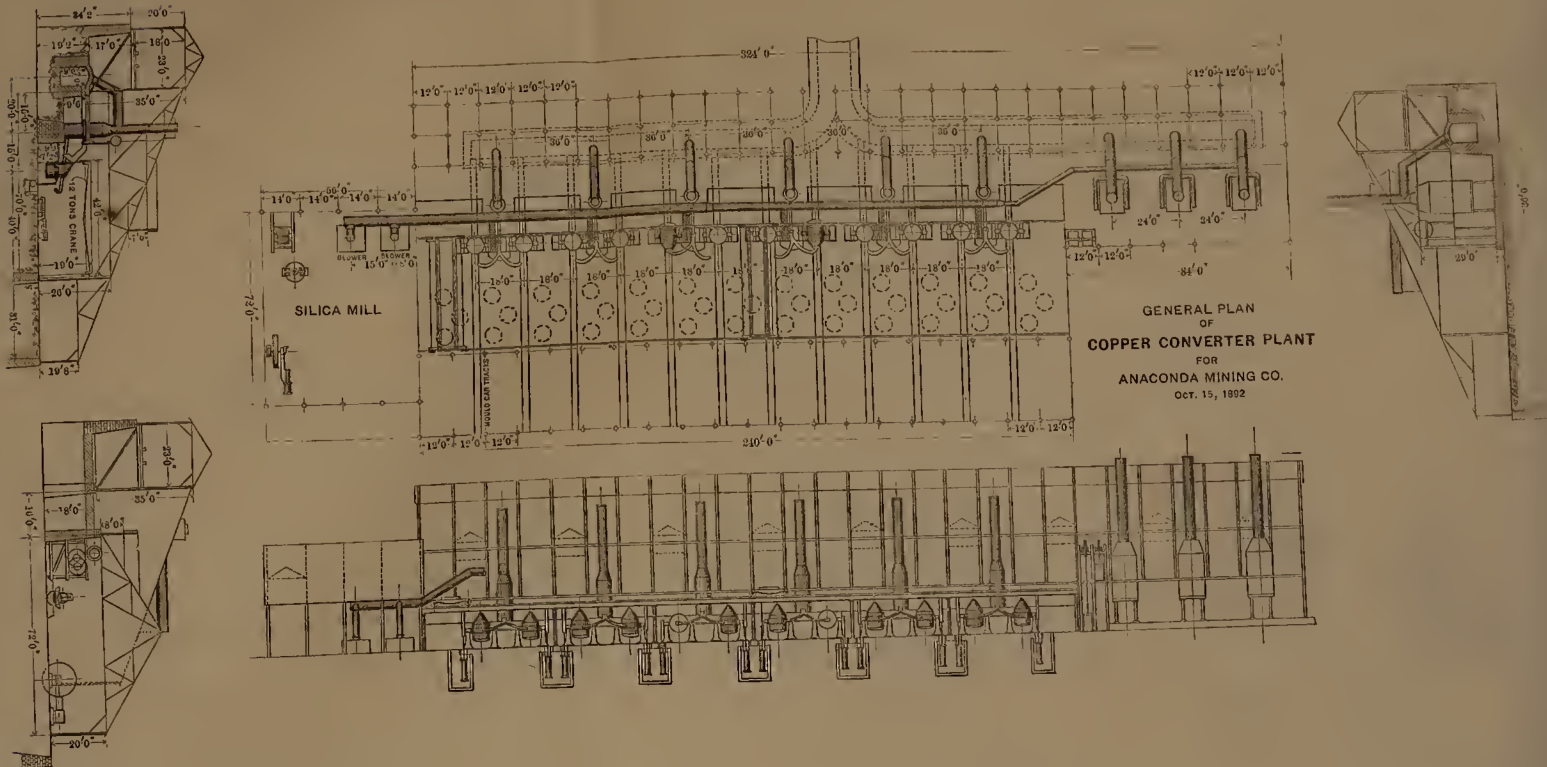


PLATE I

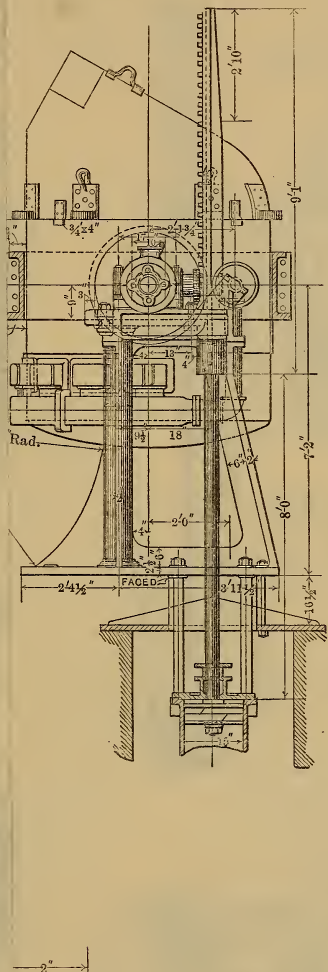


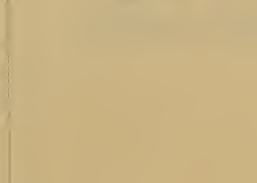
GENERAL PLAN
OF
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
Oct. 15, 1892

PLATE II



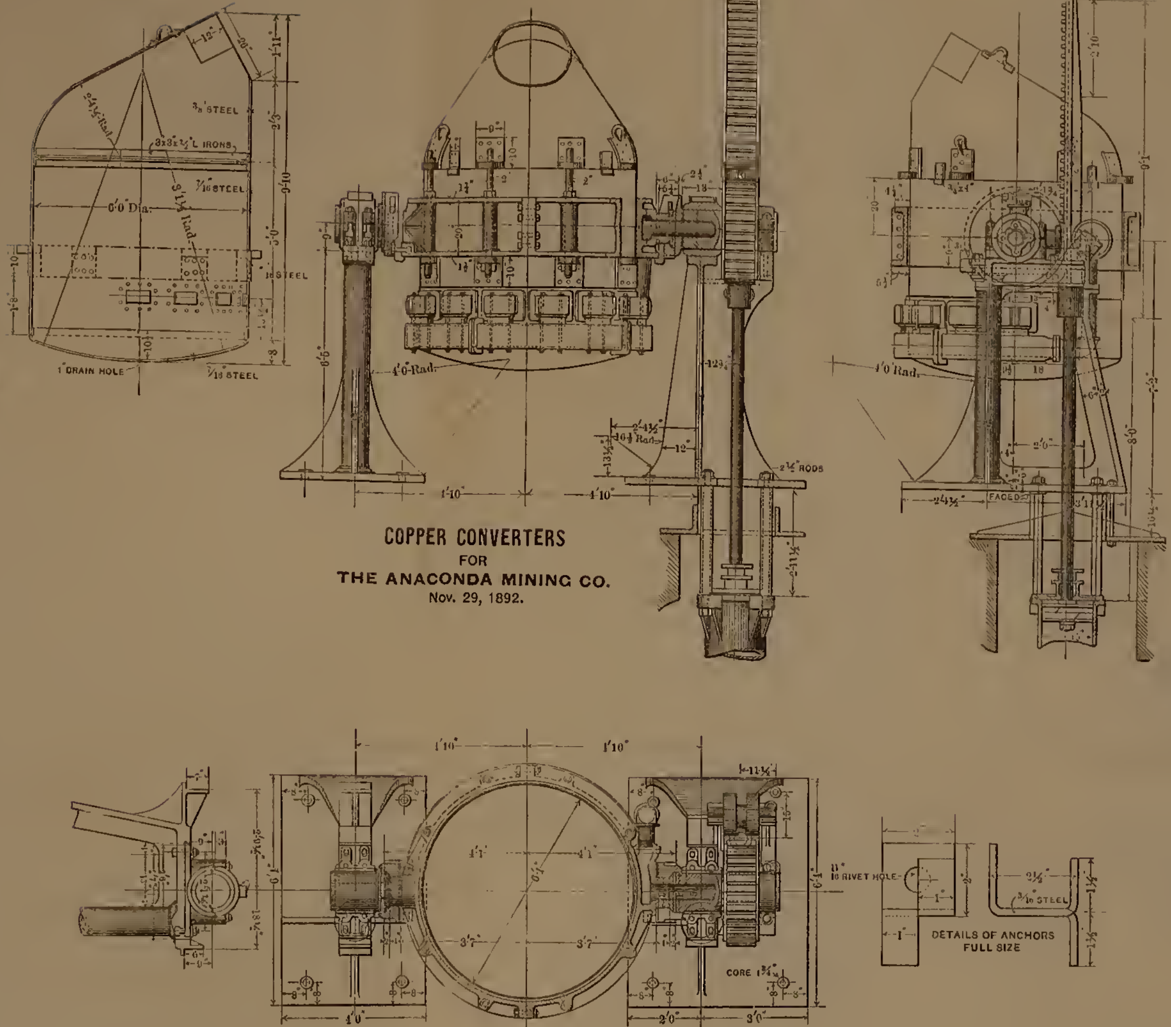
12 TONS CRANE
SILICA MILL
BLOWER
BLOWER
MOULD CAR TRACKS
GENERAL PLAN
OF
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
Oct. 15, 1892





DETAILS OF ANCHORS
FULL SIZE

PLATE III



COPPER CONVERTERS
FOR
THE ANACONDA MINING CO.
Nov. 29, 1892.

PLATE IV

DETAILS OF CONVERTERS

COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
DEC. 4, 1892.

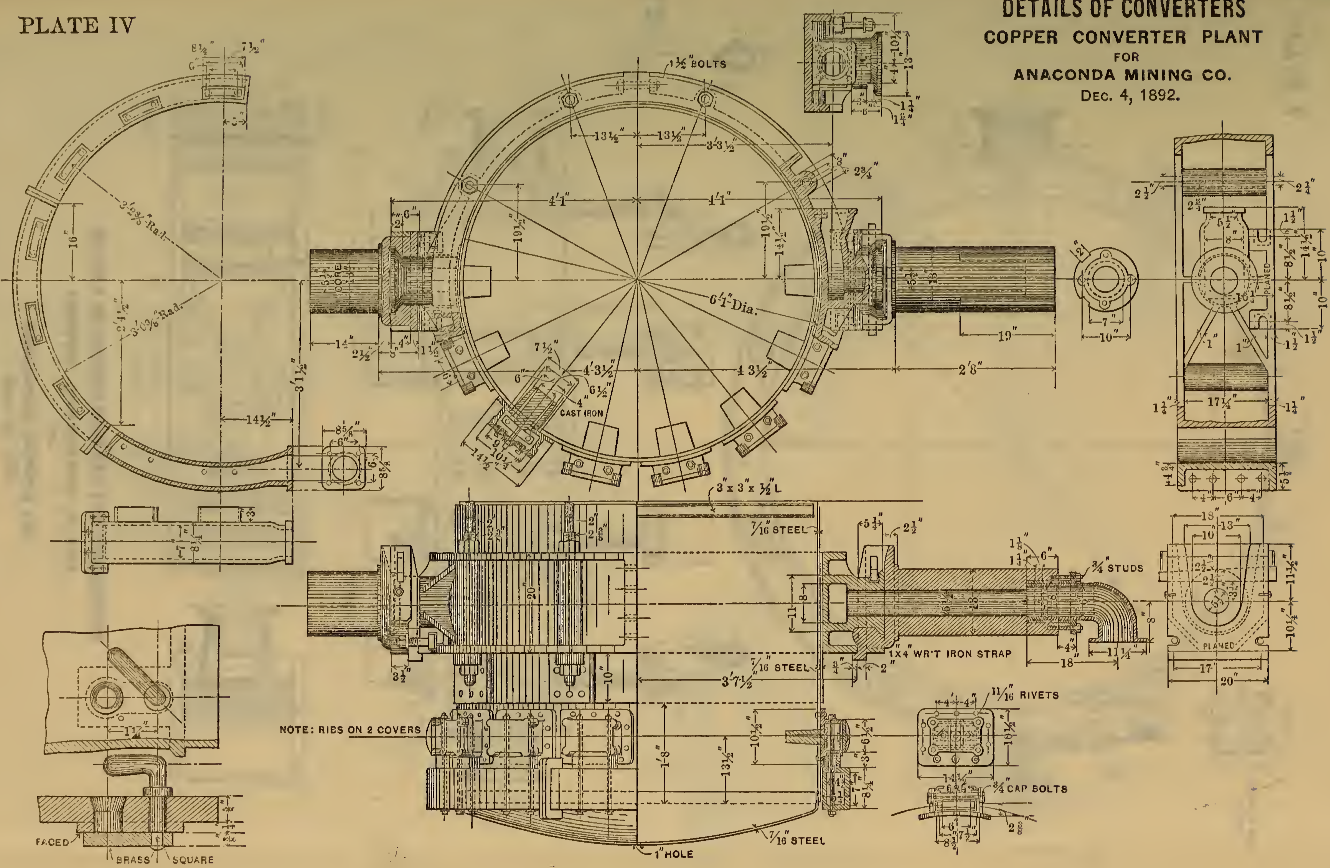
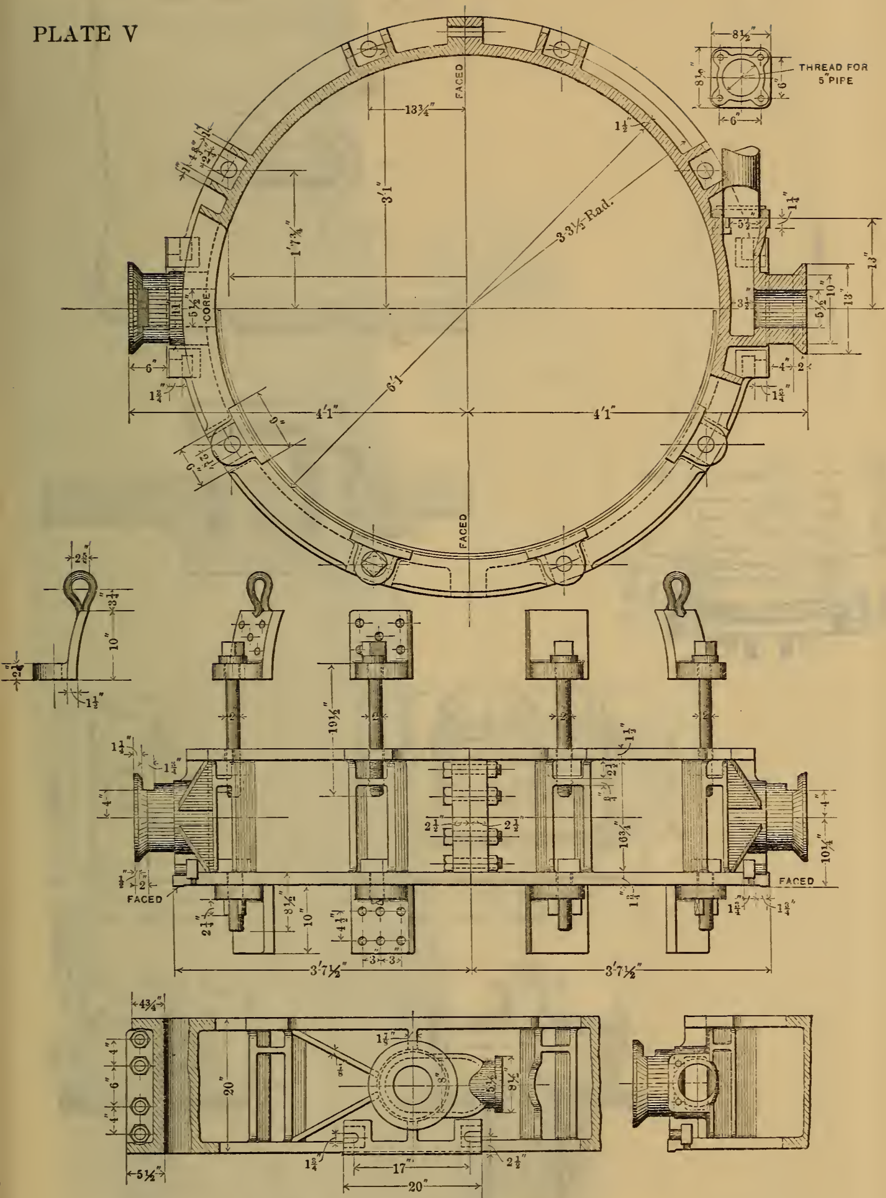
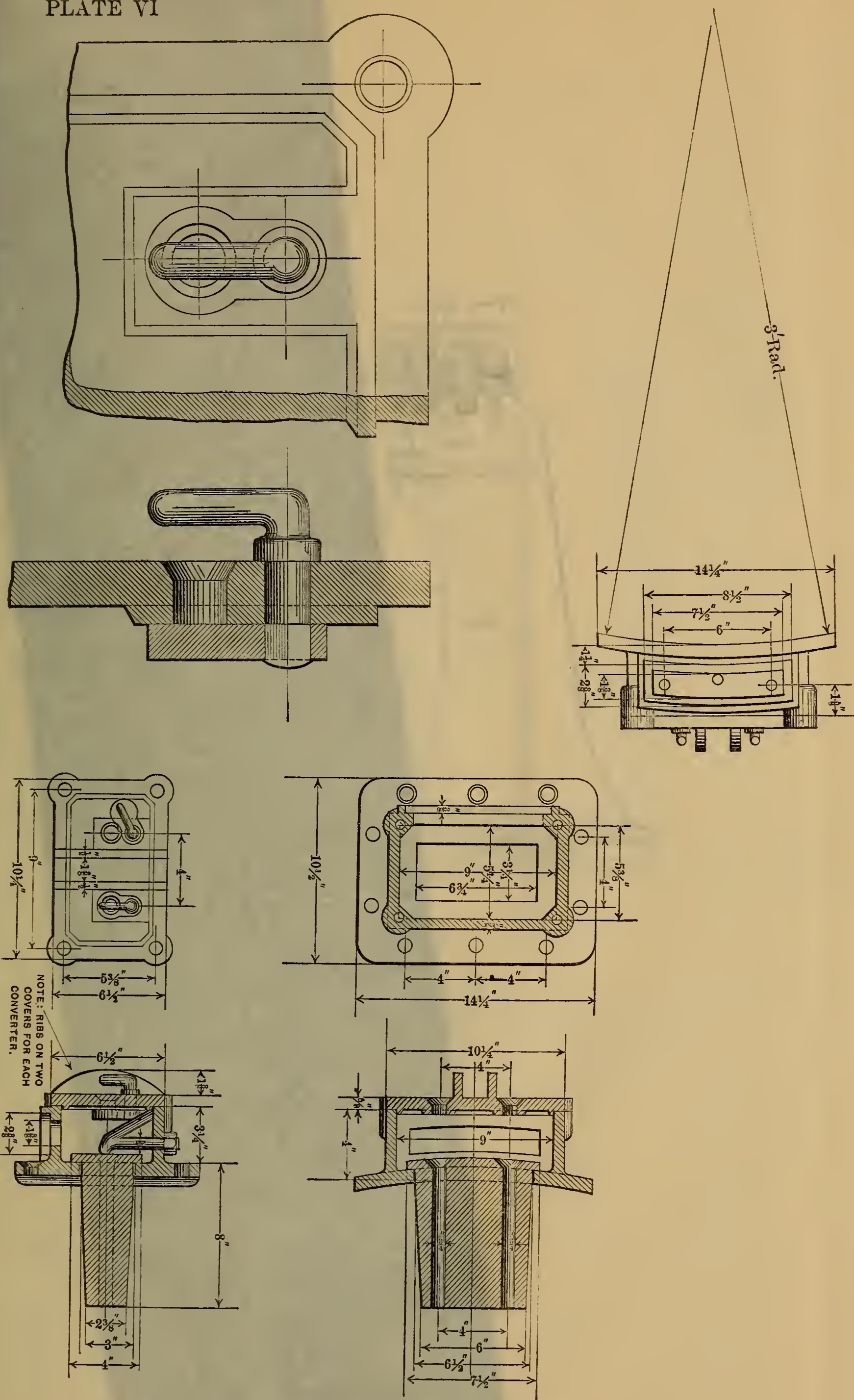


PLATE V



DETAILS OF TRUNION RING ETC. FOR CONVERTERS
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
MAR. 29, 1893.

PLATE VI



DETAILS OF TUYERBOXES AND TUYERS FOR CONVERTERS

COPPER CONVERTER PLANT

FOR

ANACONDA MINING CO.

FEB. 13, 1893.

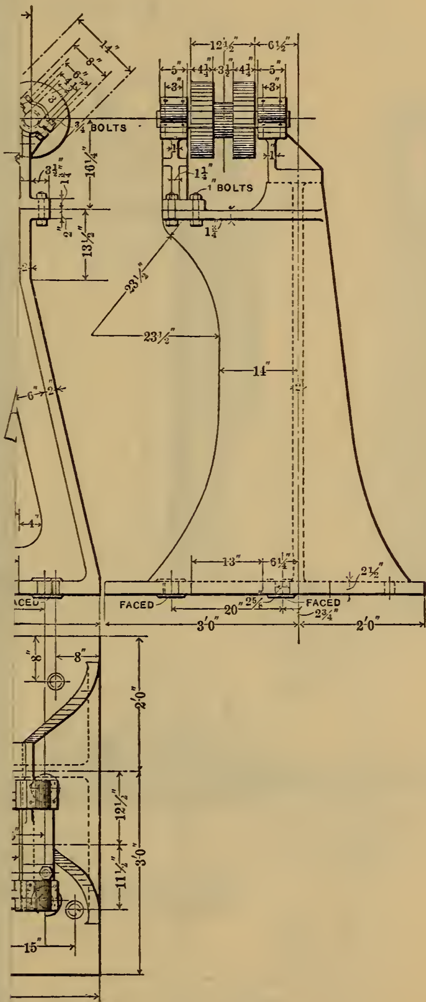
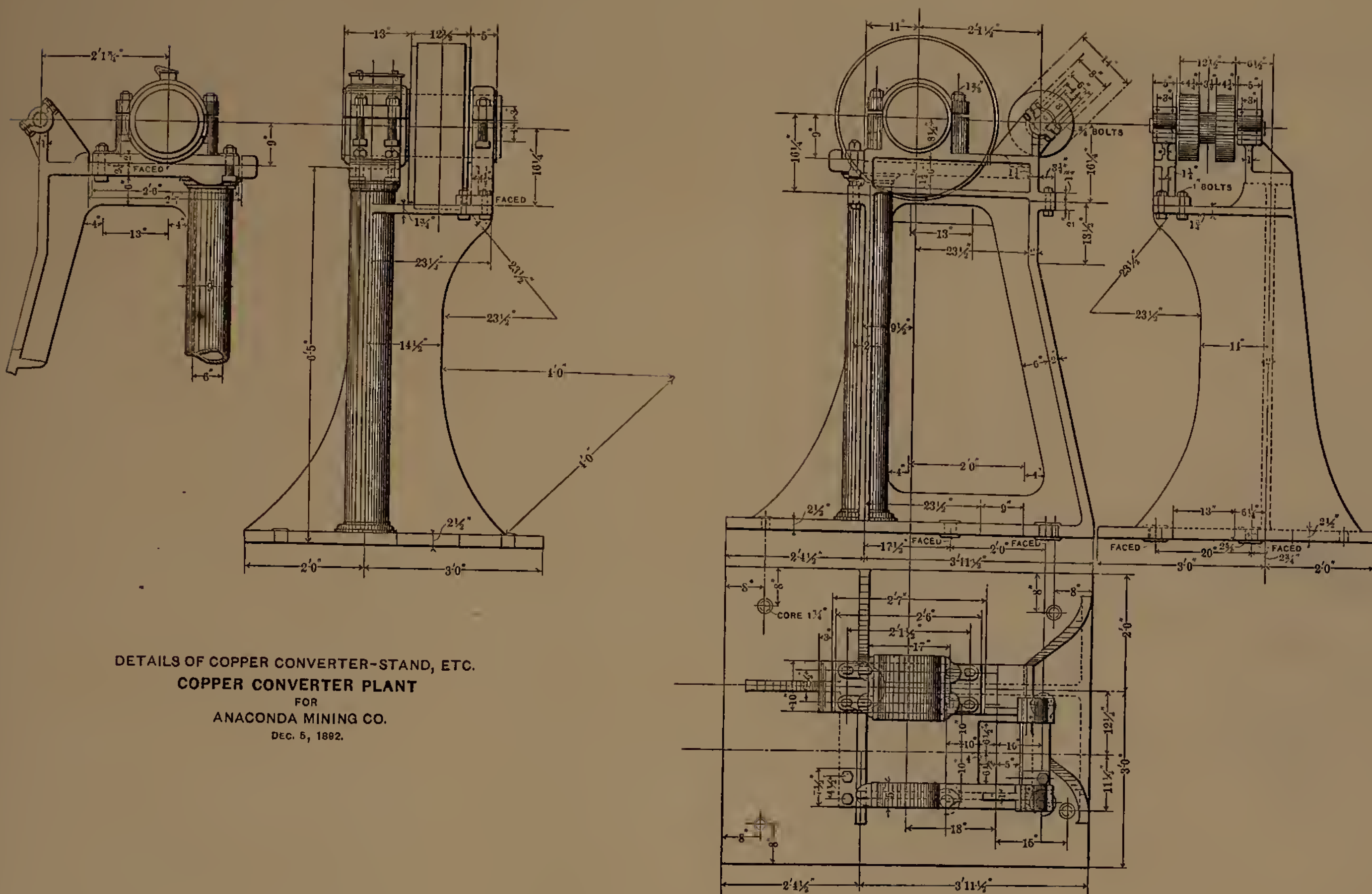
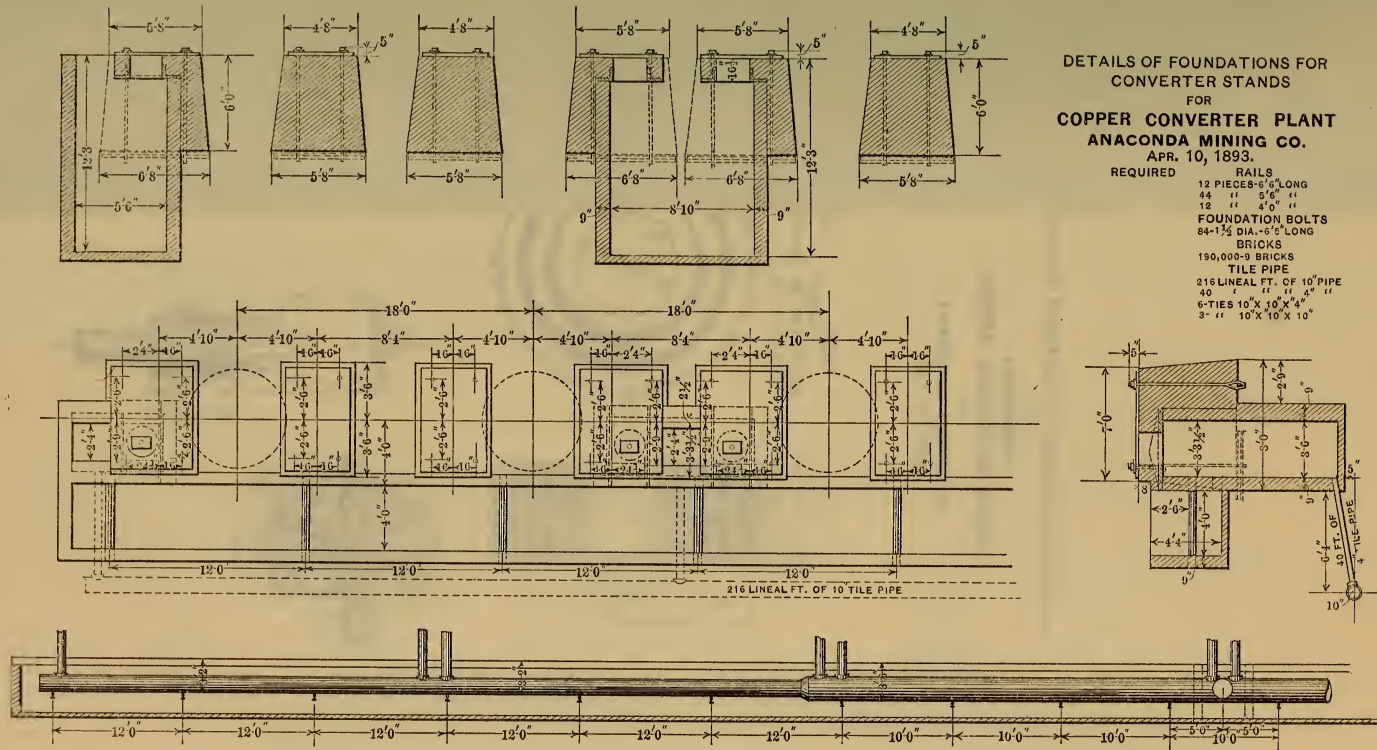

¾" BOLTS
1" BOLTS
FACED
FACED
12½"
6½"
14"
16¼"
13½"
23½"
23½"
11"
13"
6¼"
20"
3'0"
2'0"
2'0"
3'0"
12½"
11½"
15"
8"
8"

PLATE VII



DETAILS OF COPPER CONVERTER-STAND, ETC.
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
DEC. 5, 1892.

PLATE VIII


DETAILS OF FOUNDATIONS FOR
CONVERTER STANDS
FOR
COPPER CONVERTER PLANT
ANACONDA MINING CO.
APR. 10, 1893.
REQUIRED
RAILS
12 PIECES-6'6" LONG
44 " 5'6" "
12 " 4'0" "
FOUNDATION BOLTS
84-1½ DIA.-6'5" LONG
BRICKS
190,000-9 BRICKS
TILE PIPE
216 LINEAL FT. OF 10" PIPE
40 " " " 4" "
6-TIES 10"X 10"X 4"
3- " 10"X 10"X 10"
216 LINEAL FT. OF 10" TILE PIPE
40 FT. OF 4" TILE-PIPE

TAILS OF HYDRAULIC CYLINDER RACK AND GEAR

FOR

CONVERTERS

COPPER CONVERTER PLANT

FOR

ANACONDA MINING CO.

Oct. 24, 1892.

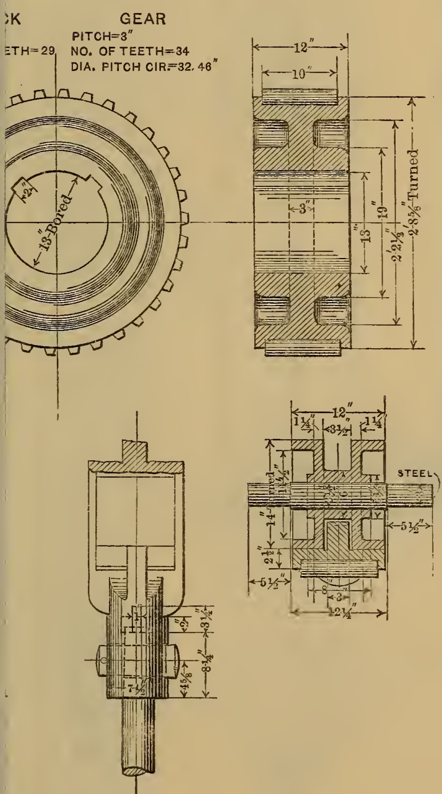
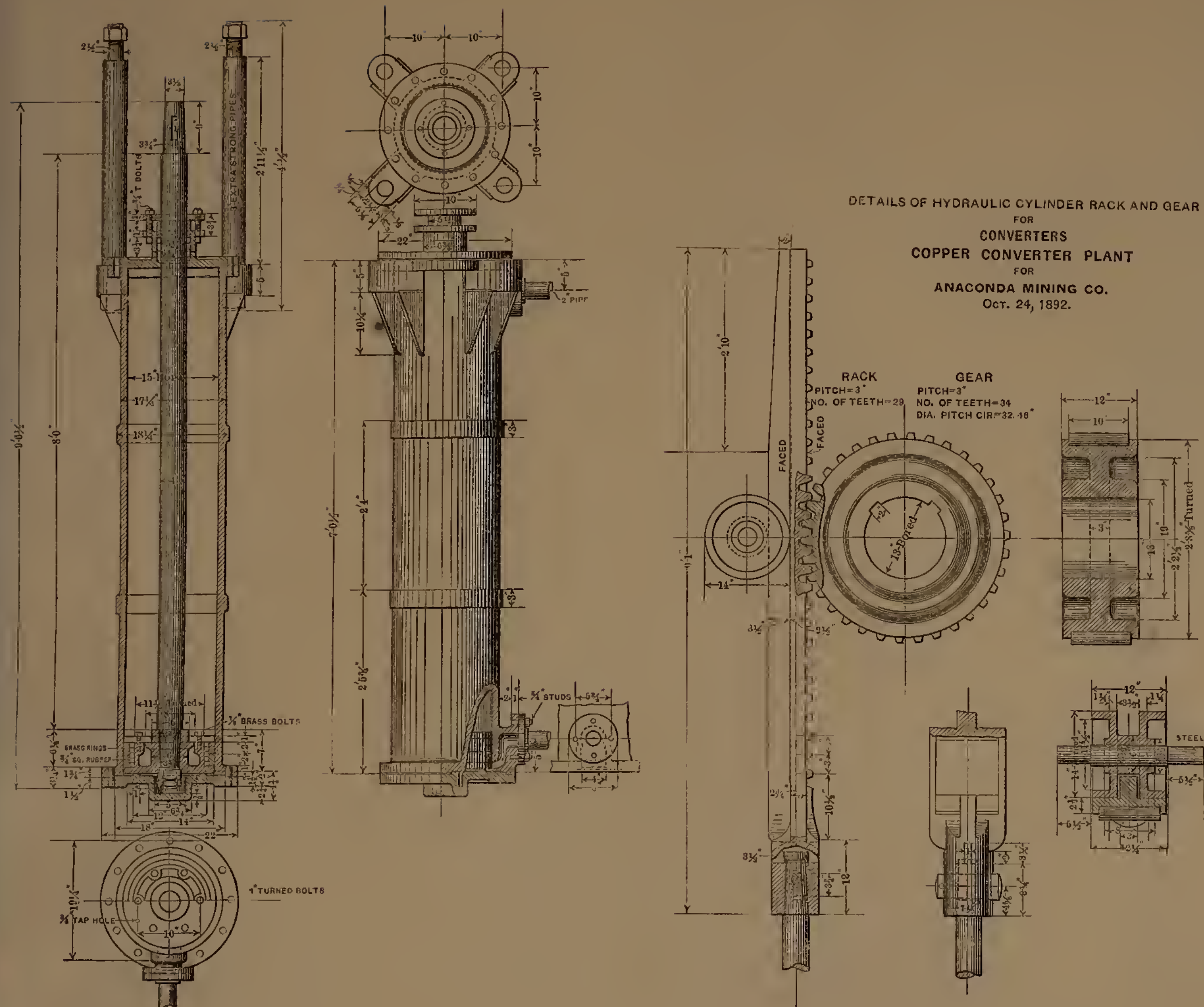
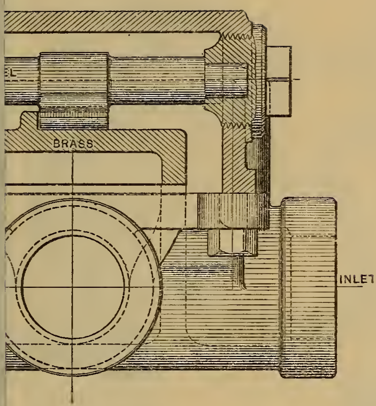


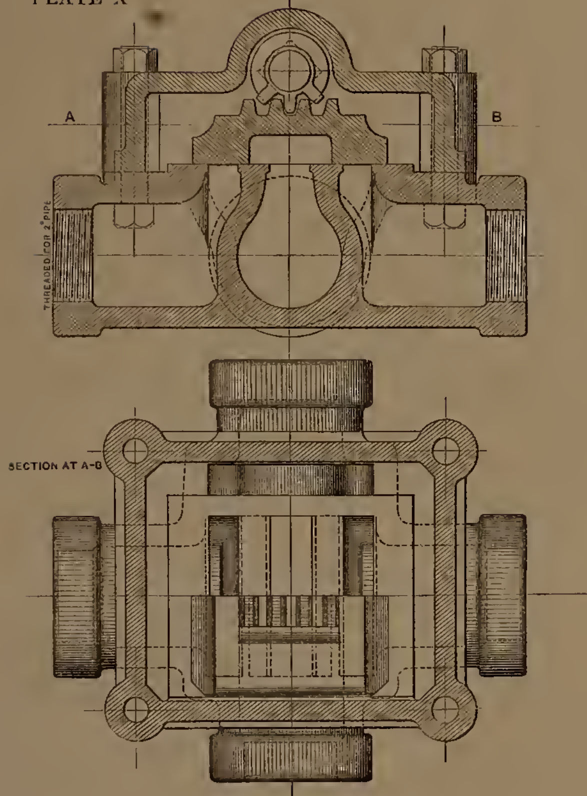
PLATE IX

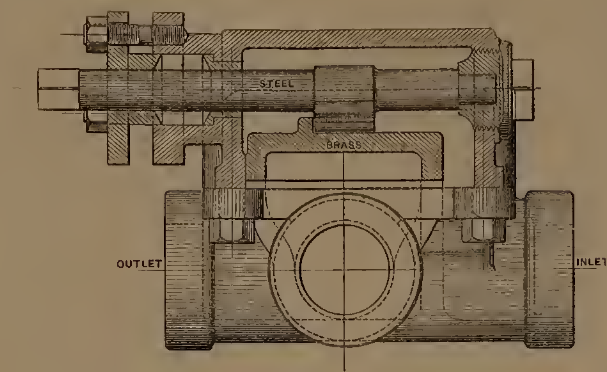

DETAILS OF HYDRAULIC CYLINDER RACK AND GEAR
FOR
CONVERTERS
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
OCT. 24, 1892.
RACK
PITCH=3"
NO. OF TEETH=29
GEAR
PITCH=3"
NO. OF TEETH=34
DIA. PITCH CIR.=32.46"
FACED
FACED
STEEL
¾" STUDS
2" PIPE
¾" BRASS BOLTS
BRASS RINGS
⅝" SQ. RUBBER
1" TURNED BOLTS
¾ TAP HOLE
EXTRA STRONG PIPES
¾" T BOLTS
18" Bored
Turned



LIC FOUR-WAY VALVE
:R CONVERTER PLANT
FOR
CONDA MINING CO.

PLATE X





HYDRAULIC FOUR-WAY VALVE
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.

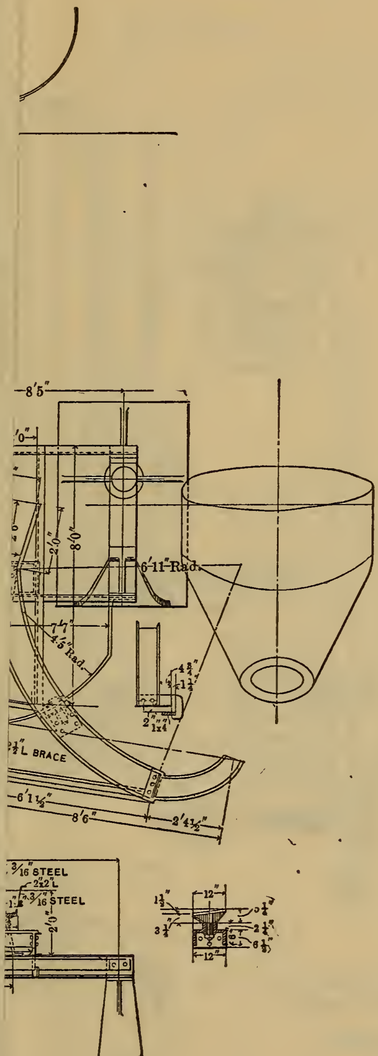
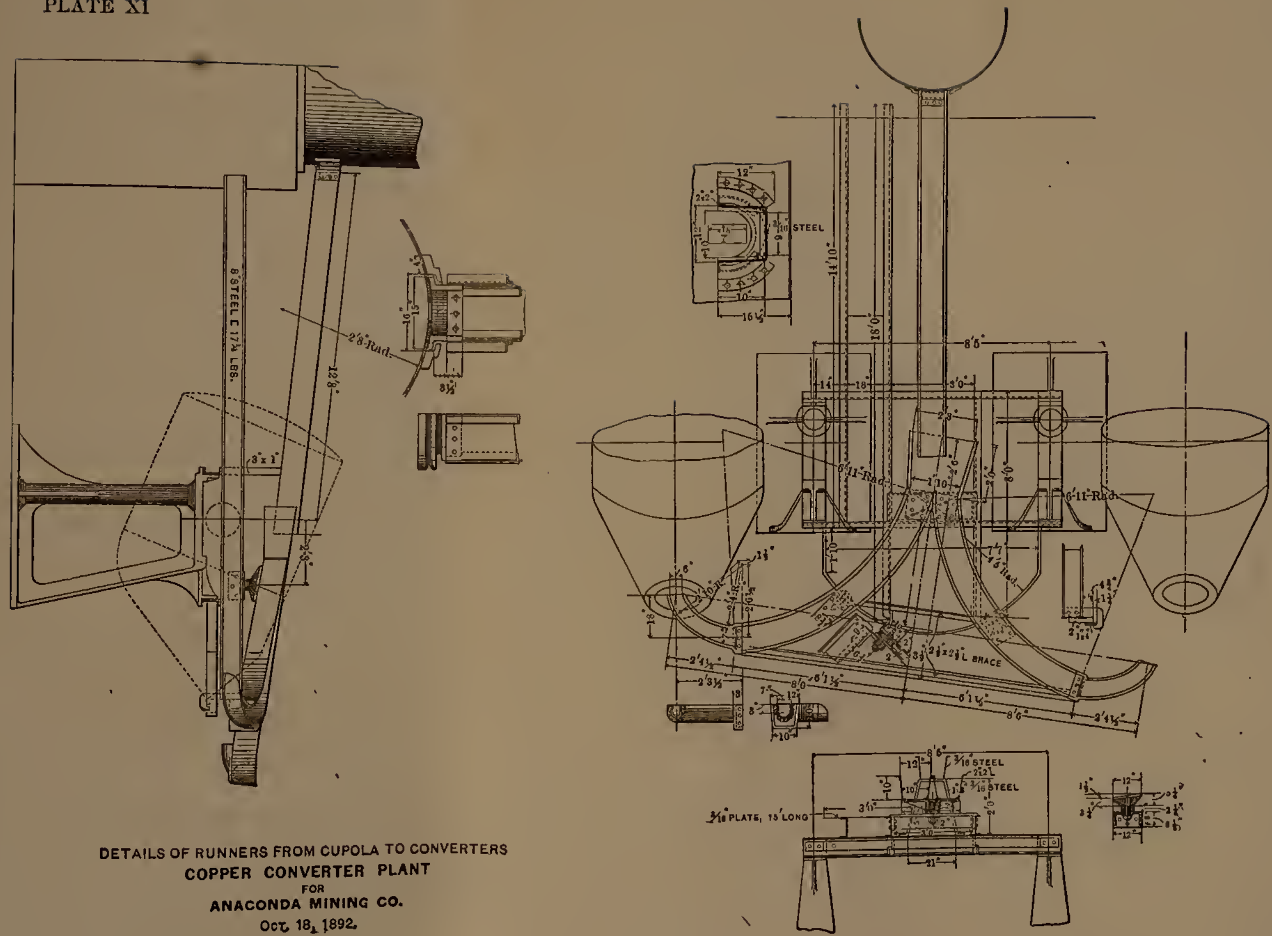

8'5"
6'11" Rad.
2'0"
8'0"
7'7"
4'5" Rad.
2½" L BRACE
6'1½"
8'6"
2'4½"
3/16" STEEL
2"x2" L
3/16" STEEL
2'0"
12"

PLATE XI



DETAILS OF RUNNERS FROM CUPOLA TO CONVERTERS
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
Oct. 18, 1892.

PLATE XII

DETAILS OF CONVERTER FLUES
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
Oct. 24, 1892.

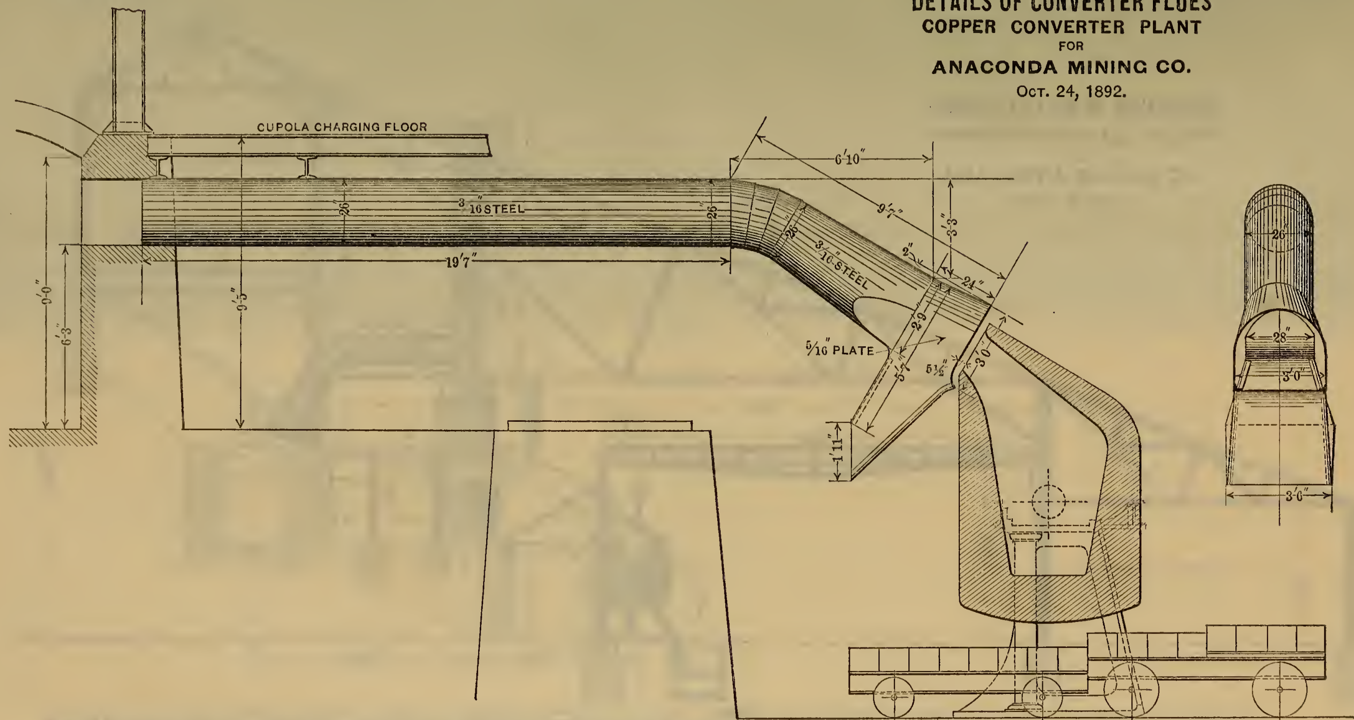
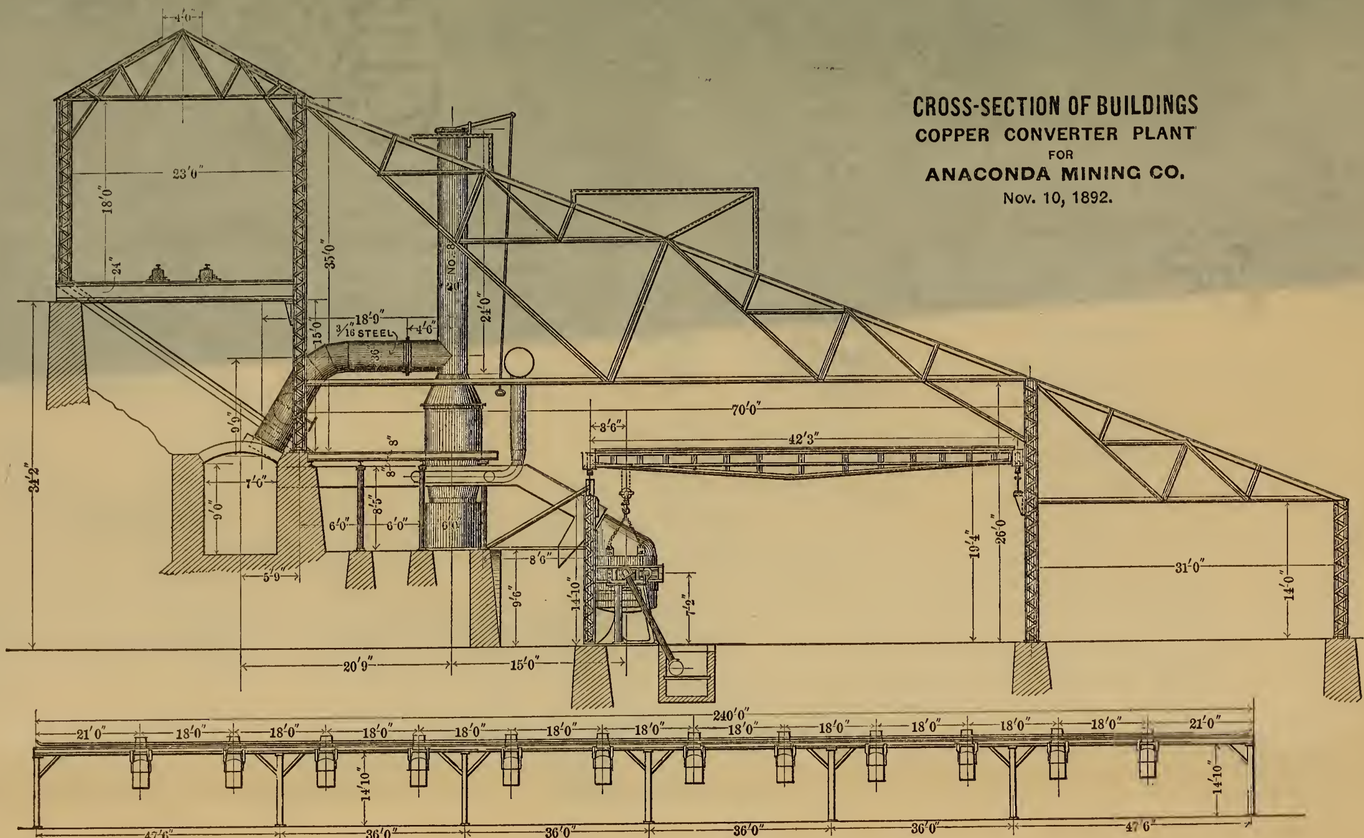


PLATE XIII



CROSS-SECTION OF BUILDINGS
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
Nov. 10, 1892.

A
T

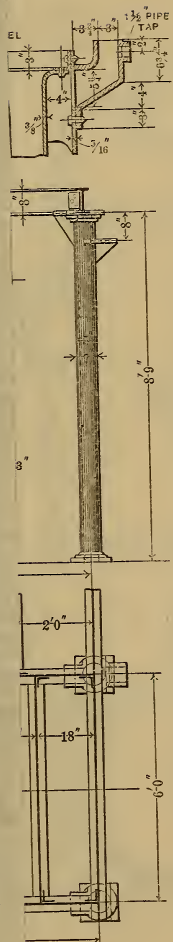
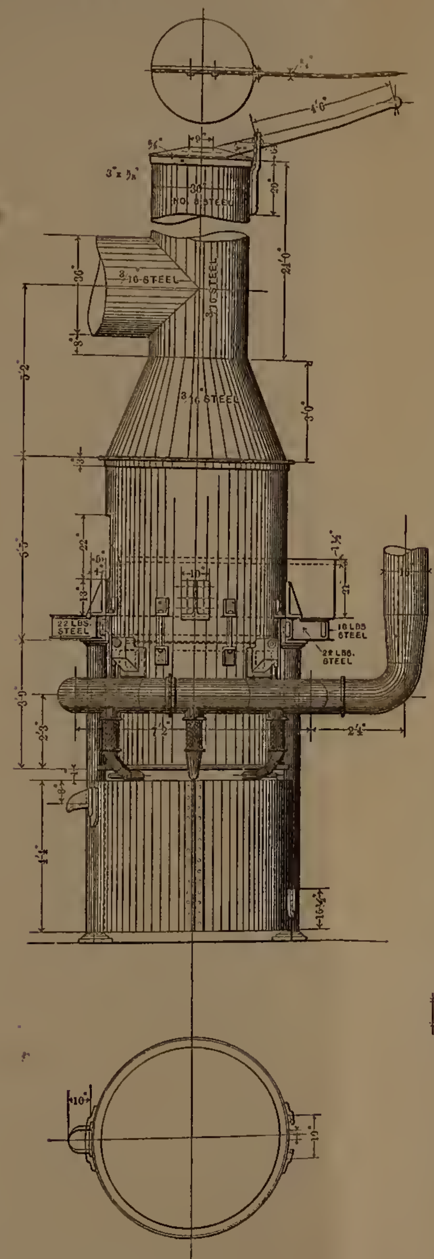
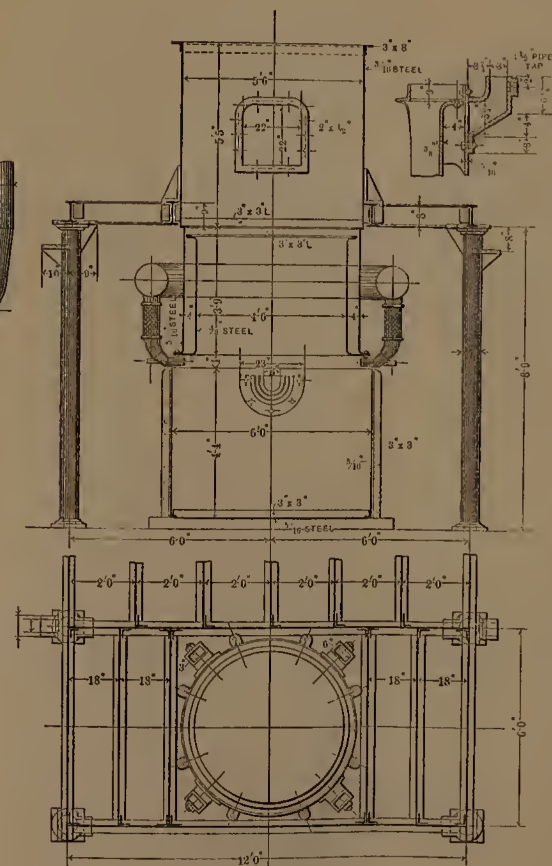
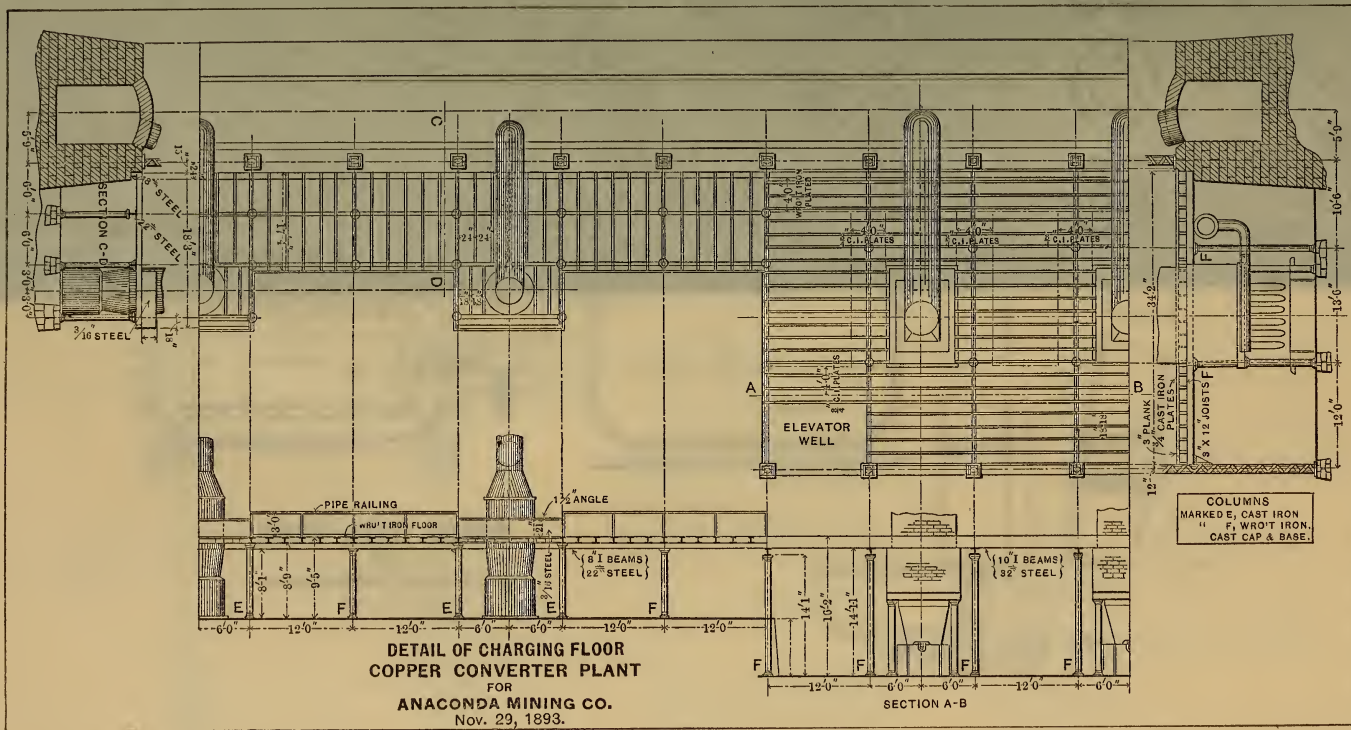


PLATE XIV



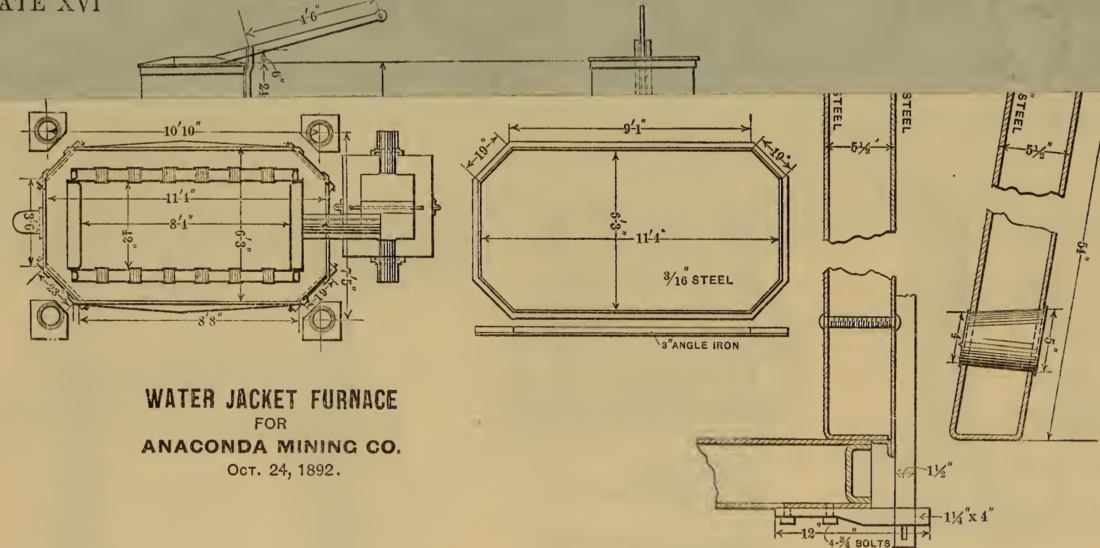
DETAILS OF COPPER CUPOLA
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
MAY 20, 1893.





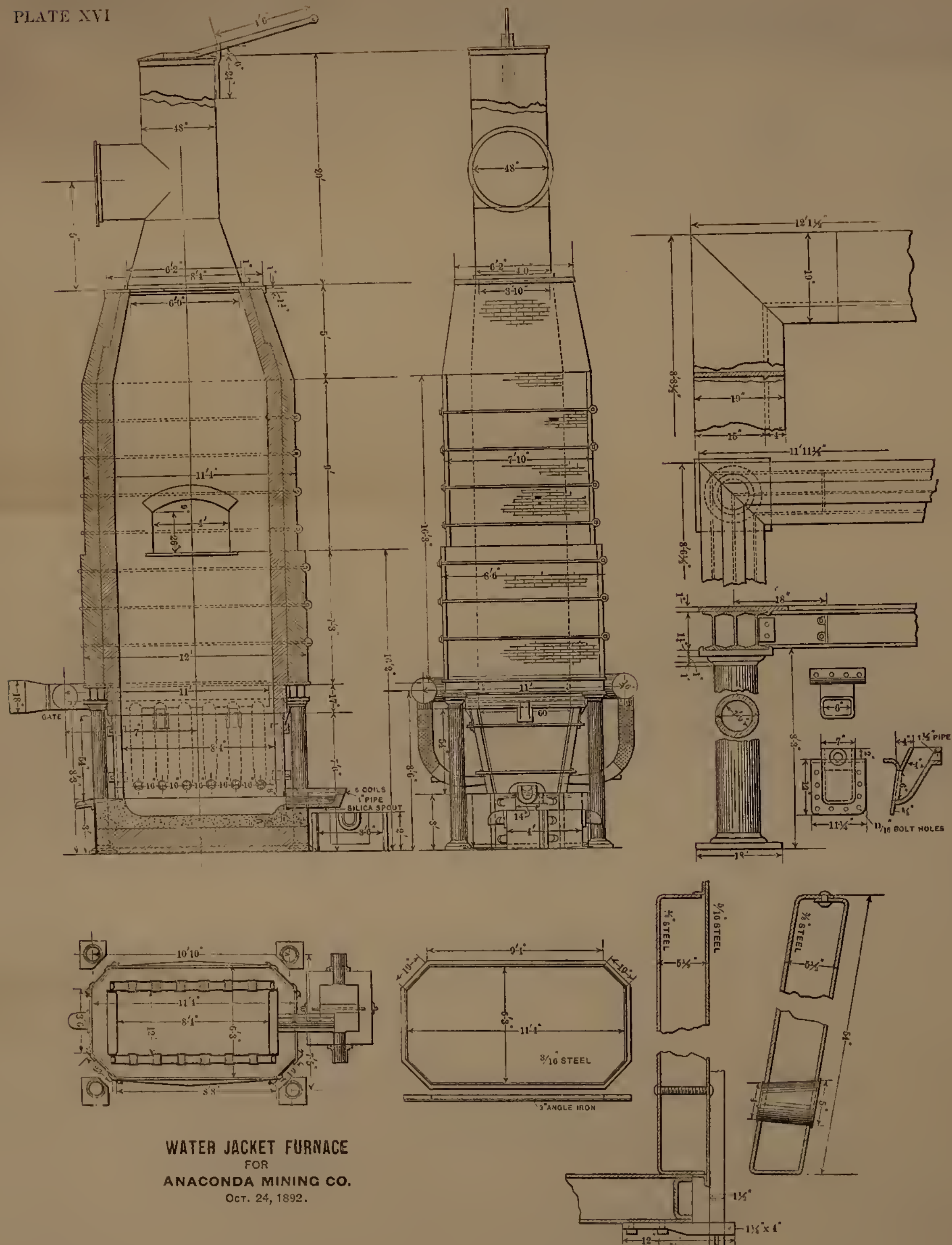
DETAIL OF CHARGING FLOOR
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
Nov. 29, 1893.

PLATE XVI



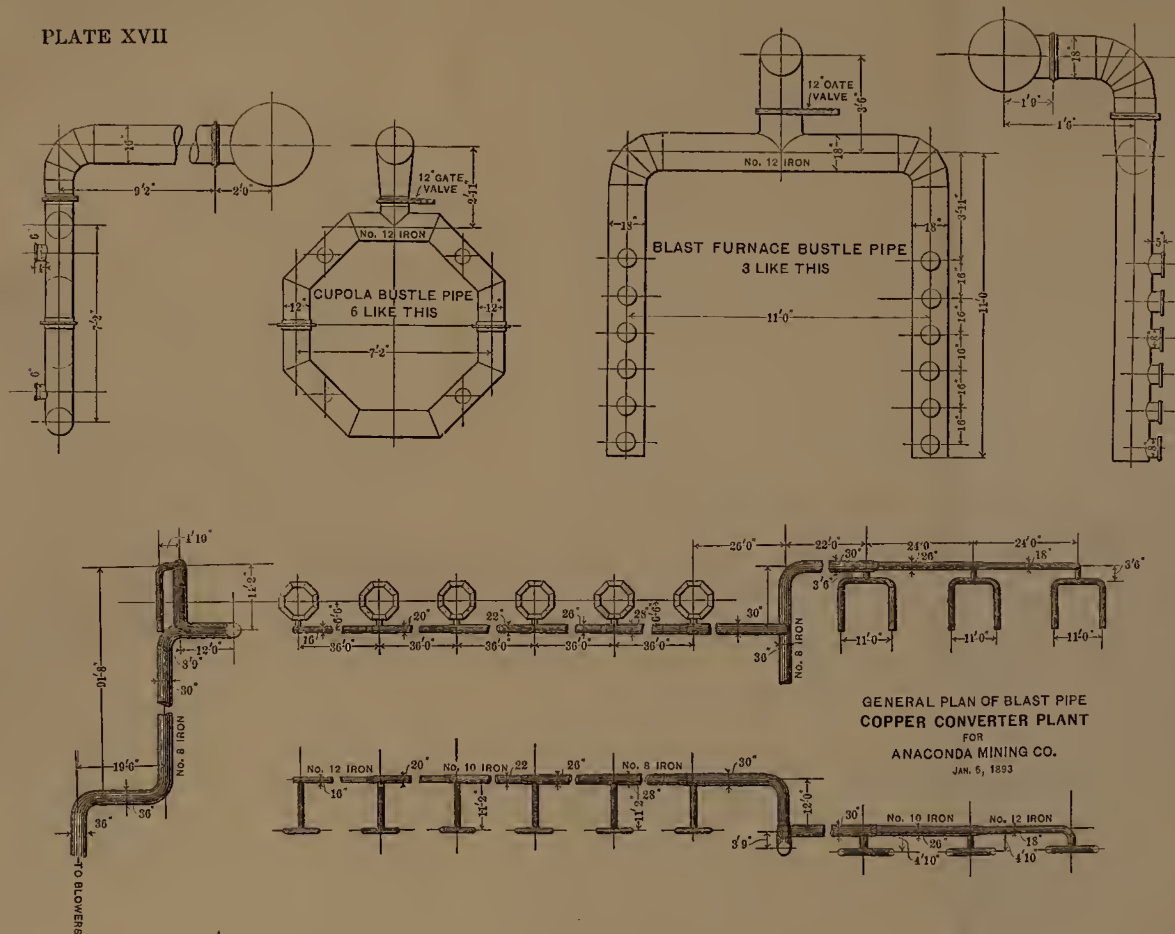
WATER JACKET FURNACE
FOR
ANACONDA MINING CO.
Oct. 24, 1892.

PLATE XVI



WATER JACKET FURNACE
FOR
ANACONDA MINING CO.
Oct. 24, 1892.

PLATE XVII



GENERAL PLAN OF BLAST PIPE
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
JAN. 5, 1893

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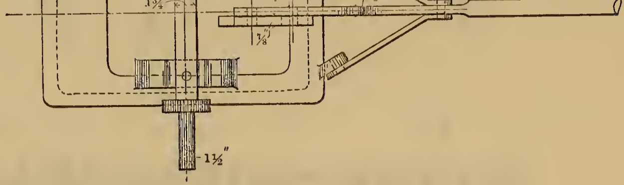

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